HANDBOOK
OF
CHEMICAL ENGINEERING

HANDBOOK OF CHEMICAL ENGINEERING

PREPARED BY A STAFF OF SPECIALISTS

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IN TWO VOLUMES
VOL. I

FIRST EDITION
FOURTH IMPRESSION

McGRAW-HILL BOOK COMPANY, INC.
NEW YORK: 370 SEVENTH AVENUE
LONDON: 6 & 8 BOUVERIE ST., E. C. 4
1922

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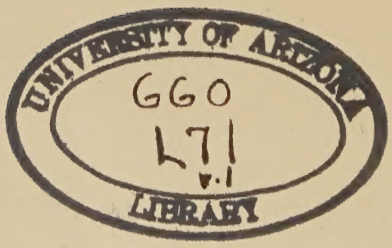
McGRAW-HILL BOOK COMPANY, INC.

PRINTED IN THE UNITED STATES OF AMERICA



THE MAPLE PRESS - YORK PA

TO

MY WIFE

THIS BOOK IS DEDICATED BY

THE EDITOR

62277

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PREFACE

If the plan of this book is not evident from the table of contents, it is as follows: An analysis was made of the chemical industries and the features essential to most of them selected for description. In this description the attempt was made to throw into prominence the basic principles, usually ending the section with a number of specific industrial applications, preferably in varied lines. The minor processes incidental or even essential to a single industry are not mentioned—there is, for example, nothing on bating for the practical tanner, nor on the production of pulp for the papermaker, though this process enters some other industries also. Where to draw the line even on what was touched was another problem. This is distinctly not an elementary text-book and a general knowledge of chemistry and mathematics is presupposed. On the other hand it is not a chemical encyclopedia, and anything that stops short of being a chemical Britannica will call down some wrathful queries as to why certain important facts were left out and why minor ones were put in. One may as well own up immediately that it was bad judgment as to the latter, and plead the publisher's unwillingness to put more white paper into the book for the former. The introduction of the matter on rare gases, radio-elements, etc., is to be justified by the newness of these subjects, and consequent lack of general knowledge concerning them. The matter of the multiplicity of contributors needs no great explanation, for we are all used to this in the modern handbooks. I believe it is a common saying that Helmholtz was the last universal genius, and we are fast arriving at the point where even a single subject becomes too vast for one man. At any rate, whether or not any of my learned confrères could write an entire chemical engineering handbook, I could not—hence the present form. I have tried not to edit the individuality out of these contributions, for I think the reader wants the writer's personality—besides, some of the contributors are large, determined, aggressive men and it wouldn't have been safe.

As to acknowledgment of indebtedness, apart from the self-evident one to the contributors, I wish especially to thank Dr. Edward Weston, E. A. C. Smith, W. Y. Westervelt, H. C. Parmelee and Percy Barbour, apart from the last's evident work as contributor, though probably no friend of mine has escaped being asked for advice on this book in the last two years.

And so, with much misgiving, this book is sent forth with the hope that chemical engineers in all lines may find something of profit in the general principles laid down, and this whether they be of the old school whose education dealt with atoms, or those of the very newest, who daily sup with electrons, or whether they be of the unhappy transitional epoch (in which stratum belongs the author) when no one fully believed the old, and no one understood the new.

New York, N. Y.
July, 1922.

Donald M. Liddell.

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HANDBOOK
OF
CHEMICAL ENGINEERING

SECTION I

POWER GENERATION AND TRANSMISSION

By W. D. Ennis[1]

Definitions.—The *foot-pound* is the unit of work. One foot-pound is the work done in raising a 1-lb. weight a distance of 1 ft. vertically against gravity. The *heat unit* (British thermal unit, B.t.u.) is also a unit of work. It is $\frac{1}{180}$ of that amount of heat necessary to raise the temperature of 1 lb. of water from the freezing point to the boiling point. One large calorie = $2.2046 \times \frac{9}{5} = 3.968$ B.t.u. One British thermal unit = 778 ft.-lb. The *horsepower* is the unit rate of doing work. One horsepower = 33,000 ft.-lb. per minute = 550 ft.-lb. per second = 2,545 B.t.u. per hour = 42.42 B.t.u. per minute. The *horsepower-hour* is a measure of work done. One horsepower-hour = $60 \times 33{,}000 =$ 1,980,000 ft.-lb. or 2,545 B.t.u. Concurrent units are the *kilowatt* and *kilowatt-hour*. One kilowatt = 1.34 hp. One kilowatt-hour = 2,650,000 ft.-lb. = 3,410 B.t.u. Registering instruments measure kw.-hr. If the work done in 5 hr. is 50,000 kw.-hr., the average load during that period was 10,000 kw. or 13,400 hp.

Load may be expressed in horsepower or kilowatts; or, for boilers and process work, in pounds of steam per hour; or as measured by consumption (pounds of coal per hour). The *load curve* is a graph of load against time. Figure 1 gives load at 2-hr. intervals for 1 day. Each ordinate is the quotient of kilowatt-hours registered in 2-hr. by 2. An *indicating* instrument would show fluctuations above and below the line. The base may be successive days in a month or year.

Average load over any time, t hr., = kilowatt-hours during that time $\div t$. In Fig. 1 it is 1,219 kw. *Maximum load* may be taken as the highest maximum shown by an indicating instrument, or (more usually) the highest average maintained during a stated period. In Fig. 1 the maximum or *peak load* maintained for a 2-hr. period is 1,870 kw. *Load factor LF* = average load $\div$ maximum load, for the stated periods. In Fig. 1, $LF = 1{,}219 \div 1{,}870 = 0.65$. It is always less than unity. The annual *LF* will be high in plants operating on a 24-hr. schedule. For electric central stations, it is usually between 0.30 and 0.40. The *LF* while running, in machine shops, may be as high as 0.50. In few industries is it over 0.60. *Rated capacity* of any machine is generally that capacity or output at which it is most efficient. If the rated capacity of this plant in Fig. 1 is 1,800 kw. it will operate at less than maximum efficiency at all loads other than 1,800 kw. *Overload capacity* = (maximum safe load − rated capacity) $\div$ rated capacity. If the plant can carry a maximum safe load of 2,400 kw. its overload capacity is $(2{,}400 - 1{,}800) \div 1{,}800 = 0.333$. *Overload* = (actual load − rated capacity) $\div$ rated capacity. For this plant, between 8 and 10 a.m. (Fig. 1) the overload is $(1{,}870 - 1{,}800) \div 1{,}800 = 0.04$ (4 per cent). *Use factor UF* or *capacity factor* = average load $\div$ rated capacity. Here $UF = 1{,}219 \div 1{,}800 = 0.68$. *Reserve capacity* = (maximum safe load − maximum actual load) $\div$ maximum actual load. If there were no fluctuations from the line of Fig. 1, this would be $(2{,}400 - 1{,}870) \div 1{,}870 = 0.28$. Reserve capacity gives a margin of apparatus to offset possible breakdowns. *Diversity factor DF* = sum of maximum demands of units com-

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posing a system, divided by maximum demand of the system. Thus assume a system consisting of three motors on which the maximum loads are respectively 100, 120 and 140 hp. The maximum load on the system will in general be less than the sum (360

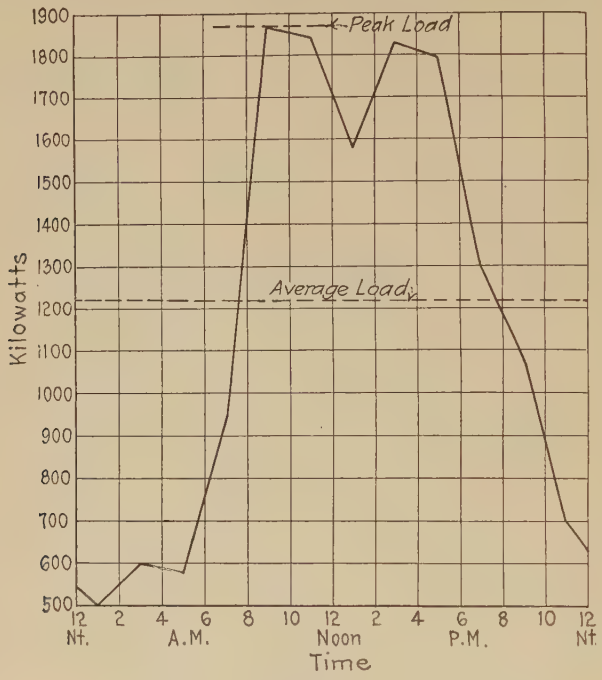


Fig. 1.—Station load curves.

hp.) of the individual maximums, because maximum loads are not apt to occur simultaneously. If the system maximum load is 180 hp. the *DF* = 2. It is desirable that it be low. The lowest possible value is 1. This will be realized when all units operate at steady loads.

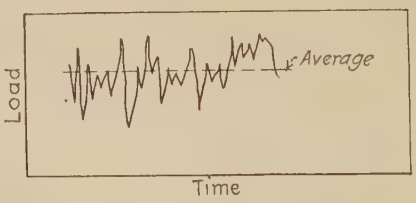


Fig. 2.—Load fluctuations.

Demand factor is the ratio of maximum demand on a system to the total rated capacity of the connected load. It is related to the diversity factor much as load factor is related to capacity factor. In the illustration just given, if the three motors are each of 125 hp. rated capacity, the demand factor is 180 ÷ 375 = 0.48. This is a most important factor in considering a power installation.

LF and *DF* depend on the load only: rated capacity and overload capacity on the equipment only: capacity factor, demand factor and reserve capacity on the load and equipment jointly.

Size of Plant.—In determining size of plant, one of the first considerations is the ratio of load fluctuations (or some load function; kilowatts, pounds of steam, pounds of coal, outdoor temperature and so on) to average load during short periods of time. In Fig. 2 the graph is such as might be obtained from a recording (indicating) instrument like an ammeter or water meter or steam-flow meter. The horizontal "average" load would be simultaneously obtained by dividing the work done (or its function) by time. Assume 50,000 kw.-hr. to be the work done in 5 hr. Then the dotted line represents the average load of 10,000 kw. The variations from this, in Fig. 2, show whether the plant capacity may safely be adjusted to suit short-period average loads (as given in Fig. 1) or whether the fluctuations must be considered. In general, overload capacities will suffice to care for reasonable fluctuations. Such "swings" as are shown in Fig. 2 are rare, when the entire plant is considered, although they may occur on individual machines. Gas and oil engines have very slight overload capacities, and if used on a load like that shown, the plant should have a capacity equal to the maximum rather than the average load.

Size of units is the next point to be decided. If a unit is to be held in reserve at all times, then the smaller the size of unit the smaller is the total investment. The size should be governed by the load curve (Fig. 1). More than one size may be used. Thus if Fig. 1 represents an average day, 700 kw. would be sufficient capacity for the period from 11 p.m. to about 6 a.m. During the time from 9 a.m. to 5 p.m., about 1,800 kw. would be needed. Units of 700 kw. and 1,100 kw. might be selected, with another 700 kw. unit in reserve, but it would be safer if the reserve unit were 1,100 kw. Some attention must be given the question of capacities commercially obtainable. The load curve should be observed for many days. Exceptional conditions like Sunday service should be considered. Provision of reserve equipment is far less the general practice than was formerly the case. In order of reliability, steam, producer gas and oil follow one another about in the order named. Waterwheels are very reliable of themselves, but the dependability of their service hangs on other factors. Purchased power from large central or transmission plants is usually highly reliable, but there may be local considerations. Reserve boilers or producers should always be provided to give opportunity for cleaning, etc.

Constant and Variable Costs.—In the illustration discussed, the total capacity of plant selected will be 2,500 kw. or 2,900 kw. as the case may be, although the average load is only 1,219 kw. Hence, the capital charges will be relatively high. The higher the load factor, the lower the rate of capital charge. For a given plant, the larger the output, the less is the "overhead" expense (interest, depreciation, taxes, insurance) per unit of output. Figure 3 shows the general law for all types of plant: the total cost is made up of a constant part plus a variable part which has a linear relation with the load. Figure 4 shows corresponding variations of cost per kilowatt-hour with load in kilowatts. Curves like those of Figs. 3 and 4 will also represent steam and coal consumption, labor cost, etc. The constant cost of these two diagrams includes not only overhead charges but also the stand-by fuel consumption when the plant is idle, non-liquid labor and maintenance charges, etc.

If the hourly constant costs are once known, and the total cost at any particular load is also known, the Klingenberg chart (Fig. 5) is useful. In the quadrant (*I*) the ordinate to any point of the curve represents the minimum load carried through-

out the entire number of hours represented by the abscissa. Hence areas under the curve represent kilowatt-hours. Thus if the line ba were straight, if its abscissa were 1,000 hr. and if the ordinates to b and a were 2,000 and 800 respectively, 1,400,000 kw.-hr. would have been generated during the 1,000 hr. of the year when the load was heaviest. Costs are laid off in quadrant (*II*): the hourly constant cost as Oc_1, the total cost when the load is a kilowatt as ma_1. Then the straight line $c_1a_1b_1$ is drawn.

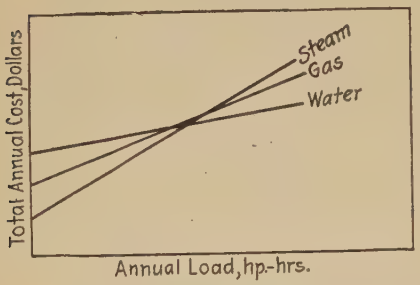


FIG. 3.—Overhead and operating costs.

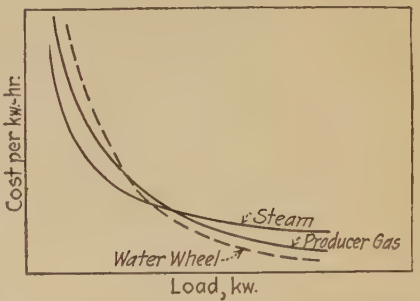


FIG. 4.—Power costs for various capacities.

When the load is b kw., the cost per hour is bb_1. Plot the curve in quadrant (*III*) a shown. Then from the point d draw a horizontal line to intersect the cost line c_1b_1 a d_1. Draw the vertical line from d_1 and continue it by a circular arc around O as a center, then draw horizontally to intersect at d_2 the vertical line from d. Then pd_2 represents the cost per hour when the load is pd. Also the area Opd_2q represents the total

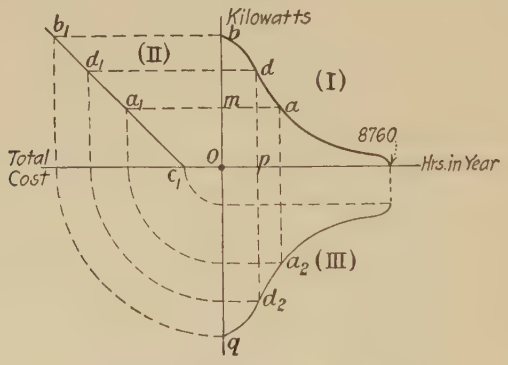


FIG. 5.—Klingenberg chart.

cost of the 1,400,000 kw. hr. represented by the area $Obdp$. During those hours of the year when the plant is heavily loaded, unit costs will be low.

These considerations may explain why, in buying power, rates may equitably vary according to the load factor proposed. A load which comes mainly at light-load periods is especially attractive to a company selling power, and it will ordinarily make low rates for such service. In all cases the effort is made to have the consumer pay his proportion of the constant costs of the generating plant, as well as those running costs which correspond with the incidence and duration of the load.

Overhead Charges.—Interest, taxes and insurance may be taken at 6, ½ to 2, and ½ per cent respectively. Depreciation may be based on the following estimated lives and residual values:

	Life, Years	Residual Value, Per Cent of Total
Buildings	10 to 40	1 to 5
Chimneys	10 to 40	0 to 10
Boilers	15 to 25	2 to 20
Steam engines	15 to 30	2 to 15
Steam turbines	20 to 30	5 to 20
Electric generators	20 to 30	10 to 25
Electrical auxiliaries	20 to 25	10 to 20
Steam auxiliaries	15 to 35	2 to 15
Gas producers	10 to 20	2 to 6
Gas engines	15 to 20	2 to 10
Oil engines	12 to 20	2 to 8
Waterwheels	15 to 40	2 to 6
Gas auxiliaries	5 to 20	5 to 8

These figures are more conservative than those often quoted, and the residual values are higher. Experience indicates that used-up machinery ordinarily finds a market at second-hand prices considerably above those prevailing for junk material. Negative residual values are possible. The life of items like foundations is limited by that of the associated equipment. If A = amount to be realized to offset depreciation = initial cost − residual value, n = estimated life, r = rate of interest, S = annual depreciation charge, then

$$S = \frac{Ar}{100} \div \left[\left(1 + \frac{r}{100}\right)^n - 1\right]$$

The blanket depreciation for the whole plant will usually be from 5 to 9 per cent of the initial cost.

Total Cost.—In steam plants, the labor cost per kilowatt-hour is usually between 0.1 and 1.0 ct., being high for small plants and low-load factors. Miscellaneous items (water, lubricants, supplies, repairs, maintenance and supervision) range from 0.05 to 0.20 ct., varying chiefly with size. These items may amount to from 3 to 12 per cent of the total operating cost. The fuel cost will usually be from one-half to two-thirds of such total operating cost. Fixed charges vary from one-half to four-fifths the operating cost, the higher ratios arising with the lower load factors.

Gas power plants involve a slightly higher labor cost, and a miscellaneous expense much higher. Fixed charges are higher than for steam because the initial investment is higher. Oil engines and waterwheels involve less labor and incidental expense than any of the other forms of power.

Purchased Power.—Rates are usually based on the purchaser's load factor: often by combining a fixed charge based on "readiness to serve" and a rate per kilowatt-hour actually consumed. Thus if the customer's peak load is 1,000 kw. and his annual load factor is 0.40, a rate of $15 per year per kilowatt for service and 2 cts. per kilowatt-hour for consumption would lead to an annual charge of $\$15{,}000 + (8{,}760 \times 0.40 \times 1{,}000 \times \$0.02) = \$85{,}080$, or the actual cost per kilowatt-hour would be 2.43 cts. If the plant should be shut down for a month, the rate would automatically increase.

Heating.—To warm average factory buildings requires about 2 B.t.u. per cubic foot per hour in the coldest weather, or about half this during average

winter conditions. If heated all day and all night for six months, the annual heat consumption will than be 4,320 B.t.u. per cubic foot, corresponding with about 4.5 lb. of steam. Some buildings require no heat. Where very slight heat is needed—as in a storehouse where it is merely desired to keep sprinkler pipes from freezing—the heat consumption is one-fourth to one-third this amount.

Process Supply.—For heating W lb. of any liquid from t_1 to t_2°F., the specific heat of the liquid being s, the heat required is $Ws(t_2 - t_1)$ B.t.u. For evaporating W lb. of water, the heat required is WL B.t.u., where L is the latent heat of vaporization at the pressure under which evaporation occurs.

BOILERS

Types and Features.—Boilers may be classed as **horizontal** (Heine, locomotive) or **vertical** (Wickes, Manning); **fire-tube** (horizontal-tubular, Manning, locomotive) or **water-tube** (Babcock & Wilcox, Wickes, Stirling, Heine); **externally-fired** (horizontal-tubular, Babcock & Wilcox) or **internally-fired** (locomotive, many marine). Internally-fired boilers (except some vertical types) are not commonly used in stationary practice. Large plants employ mostly the water-tube (approximately) horizontal form: Babcock & Wilcox, Heine, etc. Smaller plants often use horizontal tubular or vertical boilers (the latter sometimes internally fired).

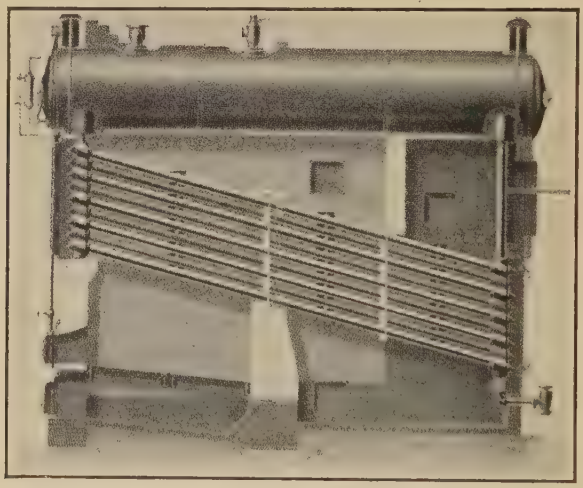


Fig. 6.—Babcock & Wilcox boiler.

Vertical boilers generally save floor space, but not if very large units are contemplated. Many such boilers have poor gas circulation and consequently are inefficient. Water-tube boilers may be shipped "knocked down." They usually contain less water, relatively, than fire-tube boilers and hence are safer as well as responsive more quickly to changes in load. Having no parts of large diameter, they are the only boilers well adapted to high steam pressures. Highest rates of evaporation are realized on water-tube boilers.

Seams exposed to the fire are objectionable in all types. The method of staying

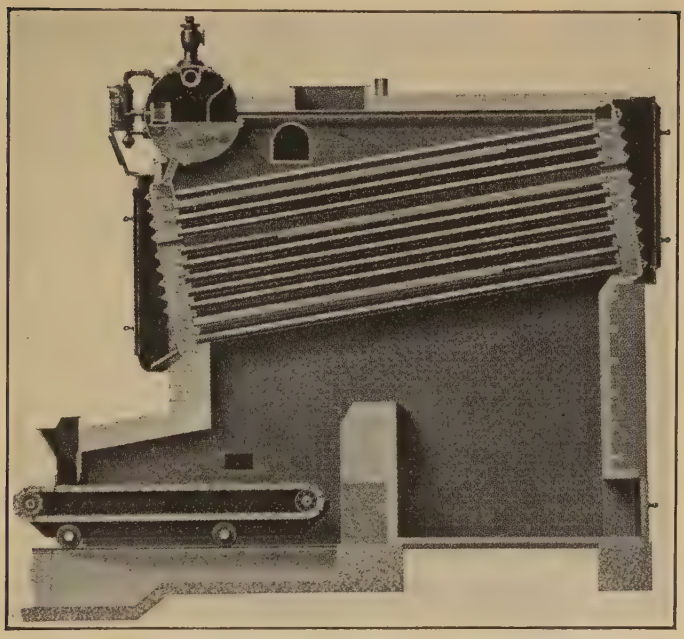
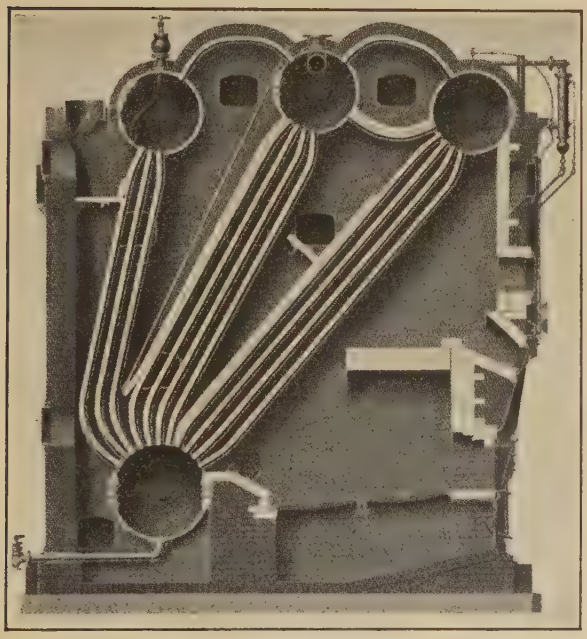


Fig. 7.—Heine longitudinal boiler with chain-grate stoker.



Fig, 8.—Stirling boiler.

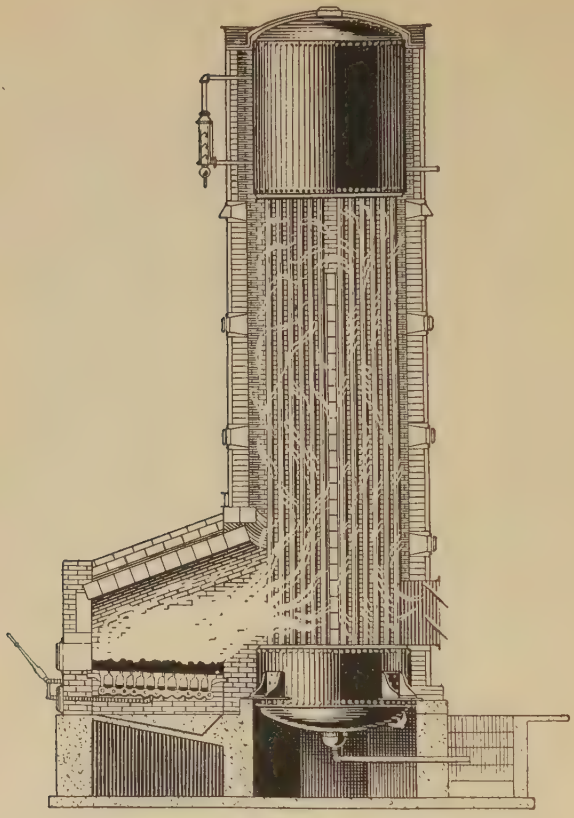


FIG. 9.—Wickes boiler.

flat surfaces should be one that will give warning of impending failure. Good circulation of water and gases is important. Water should be introduced at a cold point, high up. Gases should be repeatedly conducted across heating surfaces and gas passage areas should gradually contract from furnace to stack. Almost any type of boiler may be so proportioned as to operate at good evaporative efficiency. The Massachusetts Boiler Law and the A. S. M. E. Code are good guides for boiler construction.

Horizontal Tubular Boiler.—This is a cylindrical shell with longitudinal tubes of small diameter through which the gases pass forward after having moved under the shell rearward. A brick setting and a cast-iron front are required.

The uptake may be taken from either the front or rear of the casting, giving the "extension" or "flush" front construction. The former saves floor space. Space should be left behind boilers for access to blowoff cocks, and space also left (preferably in front) for removal of tubes and their replacement.

The thickness t (inches) of shell for a working pressure of p (pounds per square inch) is given by $t = aDp \div 2T$, where T = tensile strength of plate, pounds per square inch; D = shell diameter, inches; a = factor of safety divided by efficiency of longitudinal-riveted joint. Thicknesses are rarely below $\frac{3}{8}$ in. for small boilers or

STANDARD FIRE-TUBE BOILERS—NATIONAL ASSOCIATION OF TUBULAR BOILER MANUFACTURERS (ABRIDGED)

	Rated horsepower	Diameter, inches	Tube length, feet	3½-in. tubes		4-in. tubes		Thickness of shell, inches	Thickness of heads, inches	Grate: width × depth, inches
				Number of tubes	Total heating surface, square feet	Number of tubes	Total heating surface, square feet			
100 lb. working pressure: double-riveted longitudinal joints.	70	54	16	44	796	36	754	5/16	7/16	54 × 54
	90	60	18	54	1,079	44	1,018	5/16	7/16	60 × 60
	110	66	18	66	1,296	54	1,225	11/32	7/16	66 × 60
	150	72	18	86	1,645	70	1,546	3/8	1/2	72 × 60
125 lb. working pressure: triple-riveted longitudinal joints.	90	60	18	54	1,079	44	1,018	3/8	7/16	60 × 60
	110	66	18	66	1,296	54	1,225	13/32	7/16	66 × 60
	150	72	18	86	1,645	70	1,546	7/16	1/2	72 × 60
	180	78	18	110	2,059	88	1,904	1/2	9/16	78 × 60
150 lb. working pressure: quadruple-riveted longitudinal joints.	90	60	18	54	1,079	44	1,018	13/32	1/2	60 × 60
	110	66	18	66	1,296	54	1,225	15/32	1/2	66 × 60
	150	72	18	86	1,645	70	1,546	1/2	9/16	72 × 60
	180	78	18	110	2,059	88	1,904	17/32	9/16	78 × 60

NOTE: The rated horsepower in the above is based on an allowance of 10 to 12 sq. ft. of total heating surface per horsepower. The grate area should be varied to suit the particular coal used.

½ in. for shell diameters of 72 in. or more. Thicknesses exceeding 9/16 in. are uncommon. Efficiencies of double-strap double-riveted butt joints may be as high as 80 per cent. Girth joints may be lap riveted and high efficiency is not necessary. The usual value of the factor of safety is 5.

Heads are made ⅛ in. thicker than shells. The expanded-in tubes help to stiffen them. Additional support is provided by through braces (rods) running from head to head. Unsupported areas above or below the tubes must be further braced. Diagonal stays are preferred because they interfere least with accessibility for internal inspection. Usually five such stays are used at each end, above the tubes, running from head to shell. The allowable stress per square inch (pressure × area supported × length of stay ÷ distance from head to attachment to shell) varies from 6,000 to 8,000 lb. according to material and size. The maximum pitch at the head varies from 5¼ to 7 in., decreasing as the pressure increases and increasing with the thickness of head plate.

Tubes are 3 in. diameter (rarely less) for hard coal, 3 in. or more for soft coal. Small tubes increase the heating surface and capacity but are hard to keep clean. Lap-welded soft steel is generally used for tubes. Charcoal iron and open-hearth seamless steel are also available. Tubes are seldom over 60 diameters long: long tubes deflect and leak at the ends. They should be kept about 3 in. away from the edge of the tube sheet. Tube pitch may be about one and two-fifths times diameter, decreasing somewhat if the water is exceptionally good. Tube groups are often divided at the middle by a wide vertical gap to increase accessibility for cleaning and to improve water circulation. For similar reasons, the horizontal pitch may be ¼ to 1 in. greater than the vertical pitch. Tubes must not be carried too high in the shell, else steam room will be sacrificed and the boiler may throw moisture. A manhole (not less than 11 by 15 in.) should be provided in one head above the tubes and a handhole in each head below them. If one of the latter is a manhole, cleaning will be made easier.

Efficiencies of Riveted Joints

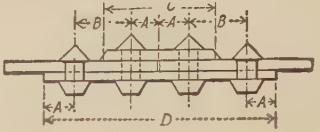


Fig. 10.

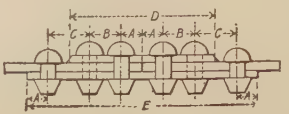


Fig. 11.

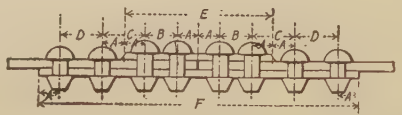


Fig. 12.

Double-riveted Butt Joints

Thickness of plate, inches	Diameter of rivet holes, inches	Efficiency, per cent	Pitch, inches		Thickness of straps, inches	Dimensions, inches					
			Long	Short		*A*	*B*	*C*	*D*	*E*	*F*
3/8	13/16	80.8	4 1/4	2 1/8	5/16	1 7/32	2 7/16	4 7/8	9 3/4		
7/16	15/16	80.2	4 3/4	2 3/8	3/8	1 13/32	2 13/16	5 5/8	11 1/4		
1/2	15/16	80.7	4 7/8	2 7/16	7/16	1 13/32	2 13/16	5 5/8	11 1/4		

Triple-riveted Butt Joints

3/8	13/16	87.5	6 1/2	3 1/4	5/16	1 7/32	2	2 7/16	8 7/8	13 3/4	
7/16	15/16	86.1	6 3/4	3 3/8	3/8	1 13/32	2 1/32	2 13/16	9 11/16	15 5/16	
1/2	1 1/16	85.8	7 1/8	3 3/4	3/8	1 19/32	2 1/4	3 5/16	10 3/8	17 1/4	
9/16	1 1/16	86.0	7 9/16	3 25/32	7/16	1 19/32	2 9/32	3 5/16	10 15/16	17 5/16	
5/8	1 1/16	86.3	7 3/4	3 7/8	15/32	1 19/32	2 5/16	3 5/16	11	17 3/8	

Quadruple-riveted Butt Joints

3/8	3/4	93.3	14	7	5/16	1 1/8	2 1/8	2 1/4	2	8 3/4	17 1/4
7/16	15/16	93.7	15	7 1/2	3/8	1 13/32	2 1/4	2 13/16	2 1/4	10 1/8	20 1/4
1/2	15/16	94.0	15 3/4	7 7/8	3/8	1 13/32	2 11/32	2 13/16	2 3/8	10 5/16	20 11/16
9/16	15/16	94.0	15 3/4	7 7/8	7/16	1 13/32	2 11/32	2 13/16	2 3/8	10 5/16	20 11/16
5/8	1 1/16	93.1	15 1/2	7 3/4	15/32	1 19/32	2 5/16	3 5/16	2 5/8	11	22 5/8

Abridged from Marks' *Handbook*. Based on tensile strength of 55,000 lb. for plates and straps and a shearing strength of 42,000 lb. (single shear) or 78,000 lb. (double shear) for rivets.

A diagonal stay or two will then be necessary and the heating surface will be reduced. The aggregate cross-sectional tube area should be one-ninth to one-tenth the grate area for hard coal and one-sixth to one-seventh for soft coal. With horizontal and vertical pitches equal to p in., the number of tubes per square foot of tubed head area is $144 \div p^2$. The distance from the center line of the boiler to the center of the top row of tubes may vary from 6½ to 10¼ in. for shell diameters ranging from 48 to 72 in. The following pitches are typical:

Tube diameter, inches	3	3½	4
Horizontal pitch, inches	4¼ to 4½	5 to 5½	6
Vertical pitch, inches	4	4½	5

The maximum practicable number of tubes is given by Parsons, as below:

Shell diameter, inches	Maximum number of tubes		
	Diameter inches: 3	3½	4
60	84	62	48
66	104	86	64
72	126	98	78
78	160	124	100

Tube thicknesses for pressures up to 175 lb. range from No. 13 for sizes under 2½ in. to No. 9 for 5 in. (B.w.g.).

Water-tube Boilers.—The Babcock & Wilcox stationary boiler is usually built of 4-in. tubes expanded into vertical or slightly inclined headers which communicate with overhead drums and a small transverse mud drum in the rear. Overhead drums are most commonly longitudinal, 24 to 42 in. diameter. The tubes incline downward from front to rear. If inclined headers are used, the header plane is at right angles with the tubes. For low pressures, headers may be of cast iron. Steel forgings are used for pressures above 160 lb. In the headers, opposite each tube end, is a handhole with inside plate and ball-head bolt. The handhole joint is either ground or provided with a thin gasket. The gases strike the tube banks transversely and usually cross them three times (three pass boiler). Heat extraction is apt to be most thorough when the tube bank is high rather than wide. This means high efficiency, a narrow boiler and low first cost of installation: but may give an insufficiently wide grate for some coals.

The Heine boiler has the gas circulation mainly parallel with the tubes, and has flat staybolted pressure surfaces forming headers or water boxes into one side of which the tubes are expanded. A particularly vigorous water circulation probably results from the use of these water boxes. A handhole is placed opposite each tube end.

Tube Thicknesses (B.w.g.) for Water-tube Boilers

Diameter, inches	Under 3	Under 4	Under 5	5	
For 165 lb. pressure	12	11	10	9	A. S. M. E. Code
For 235 lb. pressure	11	10	9	8	
For 285 lb. pressure	10	9	8	..	
For 400 lb. pressure	9	8	7 for 4 in.	..	
For 210 lb. pressure	..	..	10 for 4 in.	..	Babcock & Wilcox Practice
For 260 lb. pressure	..	..	9 for 4 in.	..	

Note: All connecting tubes (headers to drums) are No. 9 in Babcock & Wilcox boilers.

The Stirling boiler has four transverse drums, one a bottom mud drum, with bent tubes expanded directly into the riveted sheet metal drums.

All of these boilers lend themselves to flexibility of arrangement. The length of tubes, numbers in a row (horizontally and vertically), etc., may be varied to suit economical or other conditions. Boilers of this type are generally supported independently of brickwork. Standard methods of cleaning have been developed, but a large number of joints must be made after Babcock & Wilcox or Heine boilers have been completely opened for cleaning.

Boiler Steels are open-hearth. Flange or boiler steel, for general use, should have *T.S.* = 55,000 to 65,000 lb. and a yield point half this, with 25 per cent elongation in 8 in. Firebox steel, for parts exposed to fire; *T.S.* 52,000 to 63,000 lb., yield point half *T.S.*, elongation 26 per cent in 8 in. Extra soft steel for rivets, *T.S.* 45,000 to 55,000 lb., yield point one-half *T.S.*, elongation 28 per cent. Steel for stays and staybolts, *T.S.* under 62,000, yield point one-half *T.S.*, elongation 28 per cent.

Grates.—Stationary grates may have longitudinal (common bar) or round (sawdust) openings, or may in plan have singly diagonal (Tupper) or doubly-diagonal (herringbone) apertures. All are cast iron, with metal 3 to 4 in. deep at center and ¾ in. wide at top. Lengths of bar should not exceed 3 ft. The width of air space varies from ¼ in. for hard coal screenings to 1 in. for bituminous lump. The percentage of air space to total area may be as much as 70 with soft coal. Diagonal openings discourage warping but make slicing difficult. Grates should pitch slightly to the rear. Depths of 6 ft. are the maximum for convenience in hand-firing, but grates 10 ft. deep are sometimes used.

Shaking grates permit of cleaning the fire while the doors are closed. The grate should be subdivided so that a portion only need be operated at one time. Both shaking and dumping should be provided for.

Boiler Settings.—A good setting should be airtight and remain so: the repair cost should be moderate. These aims depend mainly on good workmanship, especially on firebrick arches. Independent support of the boiler proper from a structural framework is preferable to the common method of using the brickwork to carry the (horizontal tubular) boiler. In this method, lugs are riveted to the boiler shell. The front lugs rest on plates embedded in the setting walls, while the rear lugs rest on rollers which lie on plates built in the walls; but the walls at this point are built around to clear the lugs. A newly built or repaired setting should be dried slowly and thoroughly before use. A complete lining of firebrick is best. Every fifth firebrick course should be made of headers. These bricks should be kept dry until used, then laid in thin fireclay. Very hard firebrick should be avoided. The best are said to be those made from Pennsylvania clay.

The "back end" of a horizontal tubular boiler (space between rear tube sheet and inside of rear wall) should be 16 to 24 in., carefully arched over, with a support making a tight joint at the head. With the orthodox type of setting (boiler carried on walls) sound construction requires two 13-in. walls with a 2-in. air space between them. When boilers are set in a battery the 28-in. wall thus formed may serve for two adjoining boilers. Side walls are tied together by bottom and top rods passing through vertical buckstaves.

Steel casings around the entire brickwork have been found so to reduce air-infiltration and radiation as to increase efficiency from 2 to 10 per cent.

Grates should be 24 in. above the floor with horizontal tubular boilers, the shell

should be at least 28 in. from the grate: much more than this if soft coal is used. The bridge wall should come within 12 in. of the shell. The ashpit should be watertight and 1 in. below the firing-floor level, if ashes are removed by hand. Such boilers should be supported so as to pitch slightly down toward the rear. Pave the combustion chamber behind the bridge wall. Return of the gases rearward over the top of the boiler is unjustifiable: cover the top with brick or asbestos. Keep the side walls far enough apart for a suitably wide grate. Provide a large cleanout door through the setting wall at the rear.

Waste-heat Boilers.—These include boilers using waste gases from beehive coke ovens, metallurgical furnaces, brick and cement kilns, etc. They are rarely worth installing unless the average temperature of the gases is 1,200°F. or more. If it is 2,500°F., the boilers may be of standard proportions, excepting for the grate and furnace. With lower temperature, more heating surface must be used per horsepower than is normal, and the gas velocities must be increased. This last requirement implies high (usually mechanical) draft. The draft at the primary furnace must not be impaired. The evaporation per square foot of heating surface may fall to 2 lb. or so. The efficiency may exceed that of a boiler burning primary fuel, as there are no combustion losses to be considered. Dust and tar deposits may give trouble, and cleanout doors should be provided liberally. Settling chambers are often necessary. Gases containing sulphur are corrosive in the presence of moisture. Auxiliary grates, burning primary fuel, are sometimes used to steady conditions. If W lb. of gas are available per hour at T°F., and if the amount of heating surface is sufficient to reduce the gas temperature to t°F., the horsepower derived is $W(T-t)s \div 33{,}479$, where s = mean specific heat of gases.

Average Working Temperatures of Waste Gases

(Babcock & Wilcox)

Process	Temperature Fahrenheit
Brick kilns	2,000–2,300
Zinc furnaces	2,000–2,300
Copper matte reverberatories	2,000–2,200
Beehive coke ovens	1,800–2,000
Cement kilns	1,200–1,600
Nickel refining furnaces	1,500–1,750
Open-hearth steel furnaces	1,100–1,400

Sizes and Proportions.—The **heating surface** (*HS*) of a boiler is that portion of the metal surface one side of which is in contact with hot gas and having water or steam on the other side. It is measured on the fire side. For n tubes, d in. diameter and l ft. long, the tube *HS* is $\pi dln \div 12$ sq. ft. Horizontal tubular boilers contain additional *HS* in the shell (about $\frac{2}{3}\ \pi Dl \div 12$ and heads (about $\pi/288\ (\frac{2}{3}\ D^2 - nd^2)$, where D = shell diameter, in. From 8 to 12 sq. ft. of *HS* may be allowed per rated horsepower. A *boiler horsepower* is the equivalent of the evaporation of 34½ lb. of water per hour at atmospheric pressure from a feed temperature of 212°F., or $34.5 \times 970.4 = 33{,}479$ B.t.u. per hour. At 10 sq. ft. *HS* per horsepower, the rate of transmission is 3,347.9 B.t.u. per square foot per hour. The *rate of evaporation*, R_E, is the weight of water evaporated per square foot of *HS* per hour. At normal output, it is ebtween 3 and 3½. Much higher values are possible if the grate area and draft are adequate.

If H = B.t.u. in 1 lb. of steam (wet, dry or superheated) leaving the boiler, h = B.t.u. in 1 lb. of water entering the boiler, then the heat imparted to each pound of steam is $H - h$ and the *factor of evaporation* is $F = (H - h) \div 970.4$. The rate of evaporation at normal output is $3{,}347.9 \div (H - h)$. If W = pounds of water evaporated per hour, the horsepower is $W(H - h) \div 33{,}479 = 970.4FW \div 33{,}479$. If C = pounds of coal burned per hour, $W \div C$ = actual evaporation per pound of coal and $FW \div C$ = *equivalent evaporation* per pound of coal. If B = heat value of coal (B.t.u. per pound) the boiler efficiency is $W(H - h) \div CB = 970.4FW \div CB$.

The *rate of combustion* is the weight of coal burned per square foot of grate per hour. It may be expressed by $R_c = m\sqrt{d}$, where d = draft between ashpit and furnace, in inches of water, and m = 17 to 32 for anthracite (increasing with the size), 80 and 60 for bituminous run-of-mine and slack, and 50 for run-of-mine semi-bituminous coal. If G = grate area in square feet, $C = GR_c$. Also $R_c = \frac{R_e}{W \div C} \times \frac{HS}{G}$. In general, the *design ratio* $R = HS \div G$ is high for high rates of combustion of a coal of a given heat value or for good coals at a given rate of combustion. Values approaching 80 give maximum efficiency but imply a costly boiler. Values as low as 30 may advantageously be employed if the load fluctuates greatly. As a rough approximation, $W \div C = 0.0007B$.

Heat Balance in Boiler Operation (Lucke)

Distribution of heat of coal, per cent	(1)	(2)	(3)
1. Vaporization of moisture in coal	0.24	0.26	5.00
2. Radiation, etc	4.12	1.98	3.50
3. Unburned gases	0.42	2.17	1.00
4. Fuel in ash and from stack	5.50	18.11	16.00
5. Total furnace loss	10.28	22.52	25.50
6. Sensible heat of stack gases	10.73	15.19	14.00
7. Radiation, etc	4.12	1.99	3.50
8. Total surface loss	14.85	17.18	17.50
9. Total loss	25.13	39.70	43.00
10. Heat to steam	74.87	60.30	57.00
	100.00	100.00	100.00

In the above table, item (10) gives the *boiler efficiency*. The *furnace efficiency* is $100 - 10.28 = 89.72$ per cent for the first case. The corresponding *surface efficiency* is $(89.72 - 14.85) \div 0.8972 = 0.834$. Then $E_S \times E_F = E_B$ or $0.834 \times 0.8972 = 0.7487$.

Variation in Efficiency with Load.—In a given boiler with a given fuel, a straight-line law expresses within limits the relation between the heat imparted to steam and the heat actually developed in the furnace (heat of combustion of the coal, less the furnace losses). It follows that E_S decreases as output increases. If E_F, as is usual, increases somewhat for increases of load below normal, and

afterward remains constant for a reasonable increase of load, then E_B will show the characteristic illustrated, which is about normal.

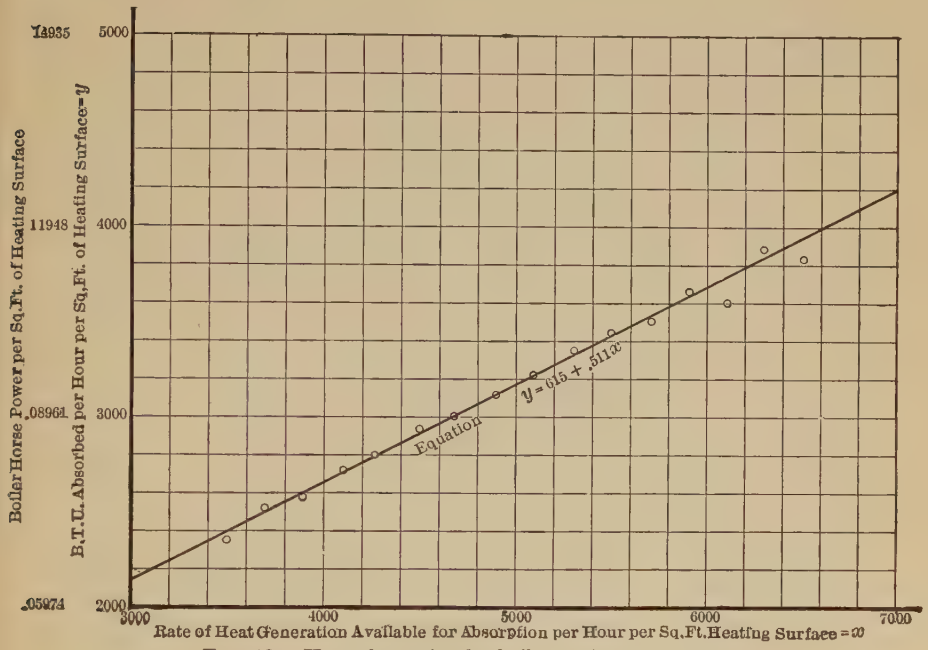


FIG. 13.—Heat absorption by boilers. (*After Lucke.*)

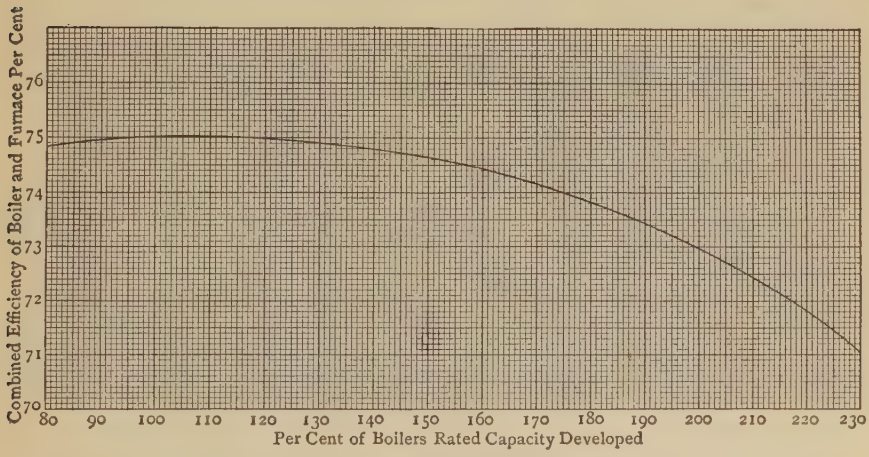


FIG. 14.—Boiler efficiency-capacity diagram. (*Courtesy Babcock & Wilcox.*)

Boiler Losses.—In the preceding table, page 14, item (4) for cases (2) and (3) is abnormally high. These losses are easily controlled. The largest and least controllable loss is that due to the sensible heat of the stack gases. If A =

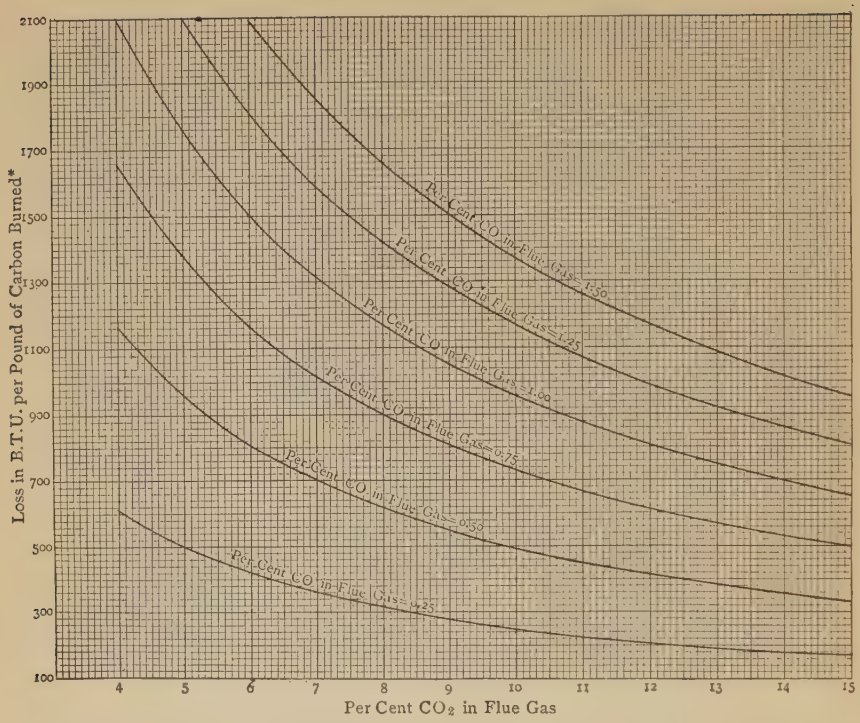

Loss in B.T.U. per Pound of Carbon Burned*
2100
1900
1700
1500
1300
1100
900
700
500
300
100
Per Cent CO in Flue Gas = 1.50
Per Cent CO in Flue Gas = 1.25
Per Cent CO in Flue Gas = 1.00
Per Cent CO in Flue Gas = 0.75
Per Cent CO in Flue Gas = 0.50
Per Cent CO in Flue Gas = 0.25
4 5 6 7 8 9 10 11 12 13 14 15
Per Cent CO_2 in Flue Gas

FIG. 15.—Heat loss by imperfect combustion. (*Courtesy Babcock & Wilcox.*)

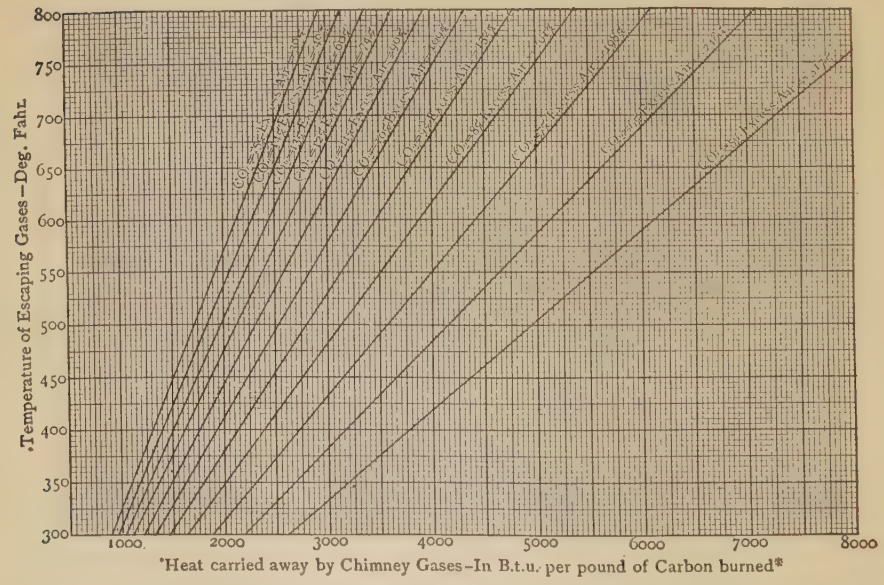


FIG. 16.—Heat lost in stack gases. (*Courtesy Babcock & Wilcox.*)

weight of air per pound of coal; T_G = temperature of gases leaving boiler; T_S = temperature of steam in boiler, T_1 = temperature of air and coal supplied, the amount of this loss, per pound of coal, is (nearly) 0.24 $(A + 1)$ $(T_G - T_1)$ B.t.u. Of this amount, 0.24 $(A + 1)$ $(T_G - T_S)$ is an *unnecessary* loss. It is reduced by keeping T_G low: *i.e.*, by using ample *HS* or an economizer: and by decreasing A close to the ideal value, which for pure-carbon fuel is 11.6 lb. Too close an approach to this value leads to other losses, due to the formation of CO.

With minimum air supply, the percentage of CO_2 in the flue gases (the amount of which is constant) is a maximum. Hence, the significance of CO_2 determination as an index of boiler efficiency: 16 per cent by volume represents remarkably good operation. The sum of CO_2, CO and O in the gases will almost invariably exceed 19 per cent. Air supply so restricted or so badly distributed as to lead to the presence of even small percentages of CO is highly wasteful: even 0.5 per cent may imply a 5 per cent loss of fuel.

Boiler Feed Waters.—Impurities in boiler waters not only reduce efficiency and capacity, but also impair quickness of response to demands for steam, increase the rate of deterioration of the boiler, and may produce dangerous conditions. The removal of deposits from a boiler nearly always involves considerable hard labor. Their prevention should be the aim in view. The unintelligent use of proprietary compounds is to be avoided. Some of them contain, besides reagents based on the water analysis, organic compounds which may loosen large masses of scale.

A water carrying 7 gr. of solids per gallon is generally considered fairly good, though much inferior waters must often be used, especially in the South and West. Even such a water will deliver 9 lb. of solids per day to a 100-hp. boiler. These solids are generally more soluble in cold than in hot water. Hence, they are deposited in the boiler, causing mud or scale. Since carbonates precipitate at 212° and sulphates at about 300°, external heating of the feed water before it enters the boiler is sufficient to remove these common impurities. External chemical treatment is more commonly employed. The most successful treatment for waters containing both sulphates and bicarbonates (or carbonic acid) is that in which lime water and soda (hydrate or carbonate) are used together. Enough soda is used to break down the sulphates of lime and magnesia. Lime is added to absorb any remaining carbonic acid. The table on page 18 represents the practice of the L. M. Booth Co.

STEAM ENGINES

Classification.—Vertical engines are uncommon, though economical in floor space and (the type being otherwise fixed) likely to be more durable. The standard mill engine is horizontal, with releasing (Corliss) valve gear, which limits its speed to about 100 r.p.m. The double wristplate engine of this type is apt to be the more economical. Smaller engines may be "high-speed" (up to 300 r.p.m.) and should be of the four-valve or automatic cutoff type. Variable speed engines are occasionally used. A link gear is preferable to a throttle-governed engine in such cases. Piston speeds are rarely above 800 ft. per minute in any type. High-speed engines have strokes about equal to diameter: slow-speed engines may have strokes two or three times as great. Large engines should have "heavy-duty" (Tangye) frames, resting on the foundation for their entire length. Condensing engines should be used where economy is important if a water supply is available. The compound engine uses considerably less

steam than the simple type: triple and quadruple engines should not be considered.

Superheated steam will lead to economy in a simple engine equal to that obtained with saturated steam in a compound. The engine and piping must be especially designed for the superheat used.

DIAGNOSIS AND TREATMENT OF FEED WATER IMPURITIES[1]

Symptom	Cause	Remedy
Priming or foaming...	Organic matter causing a scum to gather on the surface of the water	Blow down frequently. Use settling tank, filter and coagulant (alum or iron alum).
	Carbonate of soda	Blow down. If contained in water, treat with barium carbonate. If introduced as a feature of chemical treatment, change treatment.
Corrosion.....	Acids in water or formed from ingredients in water (as from salt water)	Carbonate of soda.
	Organic matter	Filter with coagulant. If from grease, use carbonate of soda. Employ mineral cylinder oils.
	Air in feed water	Exclude or liberate air. Heat feed. Add caustic soda or slacked lime. Put zinc plates in boiler.
	Chloride and sulphate of magnesium	Carbonate of soda.
	Electrolysis	Zinc plates.
Mud, sediment.....	Suspended matter	Blow down. Settling tanks, filtration.
Scale.....	Soluble salts	Blow down.
	Bicarbonates of lime, magnesium, etc.	Heat feed to 212° and filter. Use lime or lime and soda or barium carbonate.
	Sulphate of lime	Live steam feed water heating. Soda or barium carbonate.

[1] An interesting application of mass action is the use of the socalled artificial zeolites in water softening. A perfect zeolite would be $Na_2O.Al_2O_3(SiO)_2.6H_2O$. When a water containing magnesium and calcium sulphates or bicarbonates is passed through a bed of zeolites, the sodium in the zeolite is replaced by the calcium or magnesium and sodium sulphate or carbonate is left in the water. These salts do not form scale. To regenerate the zeolites a strong solution of common salt is passed the bed, reforming the sodium zeolite and magnesium and calcium chlorides. But because of the conditions governing equilibrium, the sodium zeolite cannot precipitate all of the calcium and magnesium salts from solution, nor can all of the calcium and magnesium be expelled from the zeolites by the sodium chloride. The reactions indicated form the basis of the "permutite," "refinite," "borromite" and "decalso" processes of food-water treatment. However, the reaction is complete enough for all practical purposes.—EDITOR,

Where low-pressure steam can be used for heating and process work, a simple engine may preferably be run non-condensing and even at considerable back pressure: or a specially designed compound employed and steam taken from the receiver between the cylinders. The Uniflow engine with a single cylinder gives economies (condensing) equal to that of the best ordinary compounds.

While the steam turbine has largely displaced the engine, the latter has still a distinct field. It is to be preferred where speed variation is necessary, where starting conditions are severe and for low-pressure operation of small units using saturated steam, whether condensing or non-condensing.

Size.—For simple engines,

$$I.hp. = \frac{fp_mAS}{33{,}000} = \frac{fp_mALN}{33{,}000} = fp_mK \tag{1}$$

$$p_m = P\left(\frac{1 + \log_e R}{R}\right) - p \tag{2}$$

In the above,

I.hp. = indicated horsepower to be expected from the engine,

A = average effective area of the piston, square inches,

L = stroke of piston, feet,

N = revolutions per minute (r.p.m.)

$S = 2LN$ = piston speed, feet per minute,

$K = ALN/16{,}500 = AS/33{,}000$, a constant for a given engine.

In an ordinary double acting engine, A is the cross-sectional area of the cylinder *minus* half the cross-sectional area of the piston rod. When a tail rod is used, the deduction is the whole cross-sectional area of the rod.

P = pressure at throttle, pounds per square inch, absolute,

p = pressure in exhaust pipe, pounds per square inch, absolute,

R = ratio of expansion = volume of steam when piston is at outer end of stroke ÷ volume of steam at point of cutoff.

and f is the socalled diagram factor, values of which are given below.

Pressures and Expansion Ratios [for Equation (2)].—In ordinary practice, P ranges from 75 to 250. Common values for simple engines are from 95 to 115. For non-condensing engines exhausting to the atmosphere at sea level, p is between 15 and 17. For condensing engines, p is between 1 and 2½. The value 2 should be used in design (Ennis, "Vapors for Heat Engines," 1912, p. 38). If the exhaust steam from an engine is used for heating buildings or in manufacturing processes, p may have any value from 15 upward.

The value of R is commonly around 4 in simple engines. It should increase as P increases and as p decreases, being usually between 3 and 5. It should be higher in jacketed than in unjacketed engines. The efficiency of the engine depends largely on the value chosen for R. High values of R will be adopted for an engine to be used where fuel is costly or the load steady. The overload capacity is similarly influenced: low values of R lead to high mean effective pressures—and hence to large output from a cylinder of given size—but also to low overload capacities. In general European practice cut-off is fixed so as to give $p_m = 1.2 + 0.2P$.

Values of f:

Independent cut-off, cylinder jacketed	0.90
Single valve, automatic cut-off, cylinder jacketed	0.86 to 0.88
Single valve, automatic cut-off, without jacket	0.77 to 0.82
Unjacketed throttling engines of small size and high speed	0.58 to 0.77

Size of Compound Engines.—For two cylinders developing the same power, the receiver pressure P_o(pounds per square inch, absolute) is given by $\log_e P_o = \frac{1}{2}(\log_e R - \frac{Rp}{P} - 1) - \log_e(R \div P)$, R being the maximum volume of steam in the low-pressure cylinder divided by volume of steam at cut-off in high-pressure cylinder. If A_l, A_h are the corresponding effective piston areas and $C = RP_o \div P = A_l \div A_h$, the mean effective pressures are

$$P_{ml} = (P \div R)\ (1 + \log_e C) - p$$

$$P_{mh} = P_{ml} \times C$$

and the horsepower of the whole engine is

$$\frac{fp_{ml}LNA_l}{16,500}$$

Discussion of Factors.—The value of C is usually from 3 to 4 in non-condensing and from 4 to 5 in condensing engines. It should vary directly with R. If it is made too great, the engine will, though probably economical of steam, be costly to build and deficient in overload capacity; for the maximum power of the engine, working as a compound, is obtained when $R = C$, the low-pressure cylinder receiving steam at full boiler pressure, and the high-pressure doing no work. In usual practice, the low-pressure cylinder is not built to withstand full boiler pressure, but only a lower pressure P_1, which is realized either by raising the receiver pressure to P_1 or (in unusual emergencies) by running the two cylinders as two simple engines between the pressure limits P and p and P_1 and p, respectively.

P will range from 115 to 265, preferably not under 165 for condensing engines. Values of p, non-condensing, are from 15 to 17; condensing, 1 or 2, but preferably 1. Half the tabulated values of R may be used for compound non-condensing engines. The use of jackets warrants high values for R. High values are indicated when fuel is costly or the load is steady.

Values of f may be taken as those given for simple engines with the $\sqrt{R}$ ratio of expansion.

Brake Horsepower.—The power lost in friction, f.hp., may be regarded as constant so that the brake horsepower, b.hp., is i.hp. − f.hp. at all loads. The mechanical efficiency at full load is

$$M = \text{b.hp./i.hp.} = (\text{i.hp.} - \text{f.hp.})/\text{i.hp.}$$

i.hp. being taken at full load. As the load increases, the mechanical efficiency steadily increases. At the Q proportion of full rated load, it is

$$M_1 = (Q + M - 1)/Q$$

In a double-acting single-cylinder engine

$$F = p_m ASM/S_1$$

where F = tractive or hauling force exerted, in pounds, p_m = average net piston pressure (the mean effective pressure, in expansive engines), pounds per square inch, A = piston area, square inches, S = piston speed, feet per minute, M = mechanical efficiency, and S_1 = speed of part at which the tractive force is applied—the peripheral speed of locomotive drivers or of hoisting engine drum—feet per minute.

Steam-engine Economy.—The steam rate of an engine depends primarily on the initial pressure and superheat, back pressure and ratio of expansion. The following tables refer to perfect engines (Rankine cycle). The probable

steam rate of the actual engine in average good practice may be obtained by dividing the appropriate tabular value by the appropriate one of the following factors: with high superheat, the factor is about 0.6.

	Saturated steam	
	Condensing	Non-condensing
Simple	0.4	0.6
Compound	0.5	0.65
Triple	0.6	

(From MARKS' *Mechanical Engineers' Handbook*)

STEAM INITIALLY DRY. BACK PRESSURE, 2 LB. ABS.

Ratio of expansion	Absolute steam pressure, pounds per square inch				
	70	100	130	160	215
	Steam consumption, pounds per indicated horsepower per hour				
5	13.8	13.2	12.8	12.5	....
7	12.6	12.1	11.8	11.6	11.3
10	11.8	11.2	10.9	10.7	10.4
15	11.2	10.5	10.1	9.9	9.6
20	10.9	10.2	9.7	9.5	9.1
25	10.8	10.0	9.55	9.3	8.9
30	10.5	9.8	9.4	9.1	8.7
40	....	....	9.3	9.0	8.5
50	....	....	....	8.9	8.4

STEAM INITIALLY DRY. BACK PRESSURE, 16 LB. ABS.

2	26.4	23.2	21.2	20.0	18.5
3	23.8	20.4	19.0	17.7	16.6
4	23.0	19.4	17.5	16.4	15.3
5	....	19.0	16.8	15.7	14.5
6	....	....	16.6	15.3	14.0
7	....	....	....	15.2	13.7
10	....	....	....	....	13.5

(From MARKS' *Mechanical Engineers' Handbook*)

STEAM SUPERHEATED 150°F. BACK PRESSURE, 2 LB. ABS.

Ratio of expansion	Absolute steam pressure, pounds per square inch				
	70	100	130	160	215
	Steam consumption, pounds per indicated horsepower per hour				
5	12.0	11.5	11.3	11.0	
7	11.2	10.6	10.4	10.1	9.8
10	10.5	10.0	9.6	9.4	9.1
15	10.1	9.4	9.0	8.8	8.5
20	10.0	9.1	8.7	8.4	8.1
25	9.9	9.1	8.6	8.3	7.9
30	9.8	9.0	8.5	8.2	7.9
40	....	....	8.5	8.1	7.8

(From MARKS' *Mechanical Engineers' Handbook*)

STEAM SUPERHEATED 150°F. BACK PRESSURE, 16 LB. ABS.

Ratio of expansion	Absolute steam pressure, pounds per square inch				
	70	100	130	160	215
	Steam consumption, pounds per indicated horsepower per hour				
2	22.9	19 7	18.1	17.1	16.5
3	20.8	17.6	16.1	15.1	14.6
4	....	17.0	15.1	14.2	13.4
5	....	....	15.0	13.8	12.6
6	....	....	....	13.7	12.3
	....	....	....	....	12.1

(From MARKS' *Mechanical Engineers' Handbook*)

TYPICAL STEAM RATES (LB. PER I.HP.-HR.) WITH VARIOUS VALVES

TYPE OF ENGINE	SINGLE VALVE	DOUBLE VALVE	FOUR VALVE	RELEASING
Simple non-condensing	33	30	29	26
Simple condensing	27	23	21½	21½
Compound non-condensing	25½	23	22	22
Compound condensing	20	17	15	15
Average	26¾	23⅝	21⅞	21⅛
Progressive saving, per cent	....	12	7	3

(From MARKS' *Mechanical Engineers' Handbook*)

Variables Affecting Economy.—Steam consumption is in practice reduced by high pressure, superheat, a mean value[1] for the ratio of expansion, good vacuum or low back pressure, multiple expansion (compound engine), low clearance, tight valves and pistons, good bearings and lubrication, and a valve gear which can be adjusted in each of its functions independently, while giving quick opening and closure. Wetness of steam has little influence, although mechanically objectionable. An engine need not be large to be economical. Some of the best records have been made by 100-hp. units. High speeds decrease cylinder condensation but limit the choice of a valve gear. They also require high compression, which is not economical. Every engine has a load of best economy (usually the rated load). If the load is variable, a flat characteristic (steam rate plotted against load or per cent of rating) is desirable.

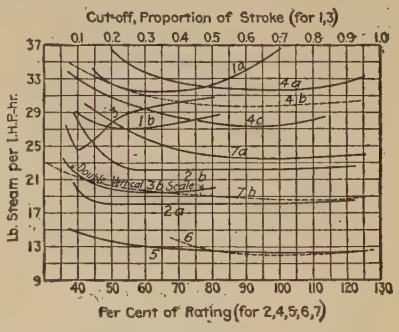


FIG. 17.—Ratio of expansion and steam consumption.

Pressures are now rarely below 100 lb.: 150 lb. is common, large central stations usually approach 200 lb. and 300 lb. is being introduced occasionally. Superheat, if used, should be at least 100°.

STEAM TURBINES

Types.—Practically all turbines are now built with horizontal shafts. The direction of steam flow (axial, vertical or tangential) is a matter of individual characteristic: axial flow is generally used in large machines. The turbine utilizes the velocity of steam. The velocity may be produced by one sudden expansion through a nozzle. This gives the *simple impulse* type, in which velocities (condensing) range upward of 4,000 feet per second. Alternately, the steam may expand (decrease in pressure) while traversing the buckets, the velocity being augmented by expansion a little more rapidly than it is expended in doing work and overcoming friction. This gives the *pressure* or *reaction* type. Impulse turbines use reaction in part, and pressure turbines employ impulse to some extent. In the pressure turbine, the velocities seldom rise above 900 feet per second. Hence the annulus filled by the blades must be large: *i.e.*, pressure turbines are large machines, usually consisting of two or more drums, each of which necessarily carries a rather large number (15 or more) rows of buckets.

Compounding.—For maximum efficiency, the peripheral speed of the wheel at the bucket pitch line should approach the value of one-half the steam speed. In simple impulse turbines, this leads either to very high rotative speeds or very large diameters. If the peripheral speed is reduced one-half, the efficiency decreases about one-third. Small capacity turbines of this type must therefore have ridiculously large wheels or else run very fast. Thus the original De Laval turbines used speeds of 10,000 to 30,000 r.p.m., with reduction gears. The present De Laval simple wheel uses peripheral speeds up to 1,300 ft. per second, implying 24,900 r.p.m. at 1 ft. diameter.

[1] Cylinder condensation makes maximum values undesirable, in spite of the superior ideal economy due thereto.

There are three methods of *compounding*, by which this characteristic is modified:

1. The steam expands in several successive *pressure stages* (nozzles), a moderate reduction of pressure and generation of velocity occurring at each nozzle. The number of rows of buckets equals the number of pressure stages. Each row receives steam at moderate velocity and may itself run at moderate speed, discharging its steam to the next set of nozzles. The De Laval multi-stage machine, of this type, uses peripheral speeds not exceeding 650 ft. per second, with good efficiency. The Kerr turbine has 2 to 13 stages with peripheral speeds from 300 to 600 ft. per second.

2. The steam is allowed to leave the buckets at high velocity, but is immediately deflected and caused to strike either another row of buckets or the same row of buckets at a new point. Several such traverses may be provided. The Terry and some of the smaller Westinghouse turbines are of this type, both using a single row of buckets. The Curtis uses several rows of buckets. *Velocity compounding*, as thus described, is probably less efficient than the use of successive pressure stages, but leads to compact and sometimes inexpensive construction. The peripheral speeds may be somewhat reduced from those of simple machines without impairing the efficiency.

3. A combination of methods (1) and (2) may be used, as in the Curtis machine which may have, say, five pressure stages with two moving wheels in each pressure stage. This leads to peripheral speeds as low as those of pressure turbines, along with high efficiency.

Size and Capacity.—The wheel diameter of an impulse turbine is determined by speeds rather than by capacity, the buckets not being filled with steam.

The nozzle dimensions (first stage nozzles in a compound machine) determine the steam flow, and thus (type and proportions being fixed) the output. Whenever the outlet pressure is less than 0.58 the inlet pressure the nozzle will have a diverging outlet. The weight of steam it will discharge is, however, determined by the *throat* (most contracted) area, the outlet area determining only the outlet pressure. The weight of saturated steam in pounds per hour that will flow through a throat area of a sq. in., is $60\ ap^{0.97} \div x$, where p and x are the absolute pressure and dryness fraction of the inlet steam, respectively. For steam superheated $T°$, the weight is less, being equal to $60\ ap^{0.97} \div (1 + 0.00065T)$.

For pressure turbines, the area of the annulus at the low-pressure end is an index to the steam flow. About two-thirds the area may be effective, the remainder being used up by the bucket thickness. The velocity to be considered is that normal to the disc, or the steam velocity multiplied by the sine of the exit angle of the buckets: say by sin 20°. The specific volume is that of wet steam at the exhaust pressure, say about 275 cu. ft. at 28 in. vacuum. Then at a maximum steam velocity of 900 ft. per second, if the diameter of the (single) low-pressure drum is d in. and the blade height (last row) is h in., the weight of steam discharged per hour is $58.8h\ (d + h)$ lb. Thus for $d = 25$, $h = 5$, the weight of steam is 8,820 lb. per hour.

Overload capacities of impulse turbines are very high. Additional nozzles may be used, or in compound machines steam at boiler pressure may be admitted to the secondary nozzles.

In addition to the impulse utilized on the forward edge of the buckets, most commercial pressure turbines are built in combination with an impulse wheel. The objectionable features of superheat, if used, are then confined to stationary nozzles: provision for overloads is readily made: and end thrust may be eliminated by causing the steam to flow both ways from a central impulse wheel through two sets of pressure elements (double flow turbine).

Turbine Economy.—The ideal steam rates, pounds per kilowatt-hour, may be summarized as follows: upper figures are for non-condensing machines at 16 lb.

absolute back pressure, lower figures are for condensing machines at 28-in. vacuum.

Steam pressure, pounds per square inch absolute	Superheat, degrees Fahrenheit					
	0	40	80	120	160	200
100	25.5	24.8	24.0	23.2	22.3	21.5
	10.5	10.3	10.0	9.8	9.5	94.3
120	23.2	22.5	21.8	21.2	20.4	19.6
	10.1	9.9	9.6	9.4	9.3	9.0
140	21.6	21.0	20.3	19.8	19.0	18.3
	9.9	9.6	9.4	9.2	9.0	8.8
160	20.3	19.7	19.1	18.6	18.0	17.3
	9.6	9.4	9.2	8.9	8.7	8.5
180	19.4	18.8	18.3	17.7	17.1	16.5
	9.4	9.2	9.0	8.8	8.6	8.4
200	18.7	18.1	17.6	17.2	16.5	16.0
	9.3	9.0	8.8	8.6	8.4	8.3
220	18.0	17.5	17.0	16.5	16.0	15.5
	9.2	8.9	8.7	8.5	8.3	8.2

The steam consumptions of actual turbines exceed these values, for the following reasons:

1. Nozzle friction, which may amount to 10 per cent.
2. Residual velocity, not utilized in buckets, causing a loss which is reduced when wheel speeds are high or several stages are used.
3. Incomplete utilization of velocity due to limitations of stream and bucket angles.
4. Bucket friction.
5. Mechanical friction and windage.
6. Electrical losses, partly constant and partly varying with the load.

Item (2) is the loss which chiefly determines efficiency. If n = number of rows of buckets, d = pitch diameter, inches, r = revolutions per minute, the accompanying curve gives values of relative efficiency, E_1, plotted against the quantity $nd^2r^2 \div 1{,}000{,}000$. The actual steam rate will then be the appropriate value from the table above, divided by E_1, and by the electrical efficiency. Any large turbine of theCurtis or modern pressure type, with the usual superheat and a good vacuum, may be expected at rated load to use not over 16 lb. of steam per kilowatt-hour. Non-condensing turbines and small turbines are inefficient, the latter in part because of the necessary simplicity of construction. High vacuum is particularly desirable in turbine installations. Superheat not only pays thermodynamically, but also decreases erosion and steam friction. Figure 19 shows the "flat" type of characteristic to be expected with a good machine. Turbines generally are differentiated from reciprocating engines by low friction losses, large potential capacities and high-rotative speeds.

Low-pressure Turbines.—The capacity of an existing reciprocating engine plant (even though operated condensing) may be increased from 60 to 100 per cent by the appropriate installation of a turbine receiving steam exhausted by

the engine at or near atmospheric pressure. Some additional boiler capacity will be needed. This large gain is due to the better utilization by the turbine of low pressures. A good vacuum is assumed. The turbine will be particularly

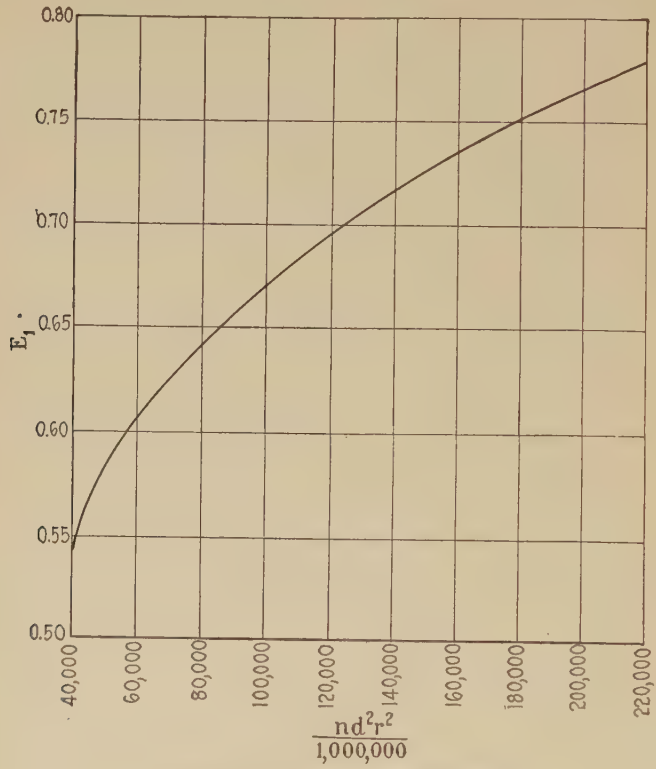
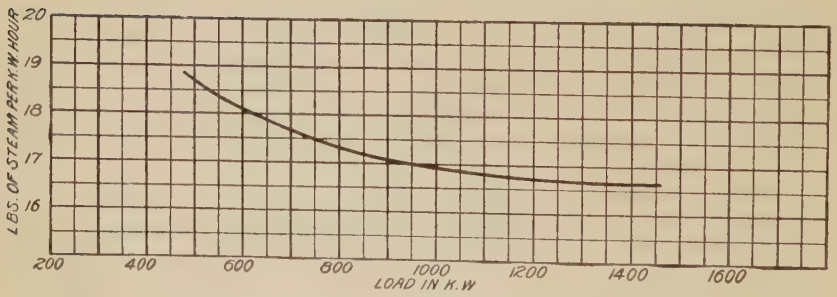


FIG. 18.—Turbine efficiency curve.



1,500 r.p.m., 4 stage, 170 lb. gage, 170° superheat, 28 in. vacuum.

FIG. 19.—Load-water rate curve for 1,000 kw., Curtis turbo-generator.

efficient because of the low steam velocities which it handles. At the Interborough plant in New York the engine was of 7,500-kw. capacity, using saturated steam at 190 lb. gage pressure, at the rate of 17.82 lb. per kilowatt-hour. The addition of the low-pressure turbine reduced the rate to from 13 to 14 lb. per

kilowatt-hour: the vacuum, however, was concurrently increased from 25 to 28 in. The diagram shows the economy curve of a low-pressure turbine used with a regenerator (equalizer and storage tank for the steam), having an inlet pressure of 15 to 16½ lb. absolute with 25- to 27-in. vacuum.

Extraction Turbines discharge some of the steam from between pressure

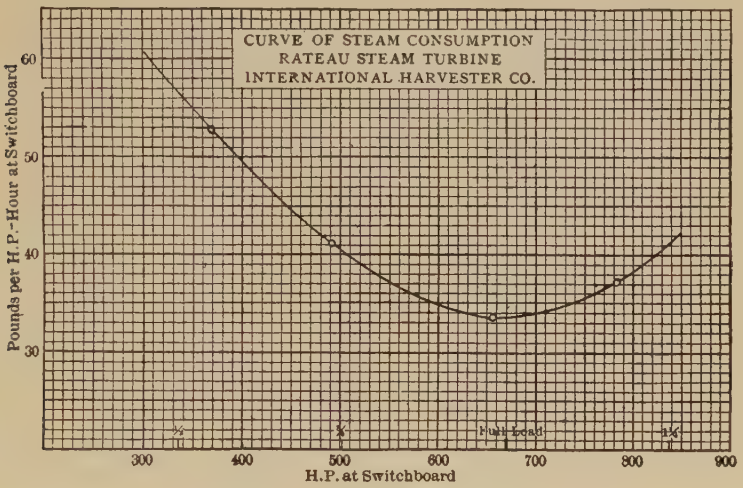


FIG. 20.—Turbine economy curve.

stages for heating and process supplies. The arrangement is analogous to the drawing off of steam from the receiver of a compound engine (page 19).

STEAM PLANT AUXILIARIES

Pumps.—Boiler-feed pumps are of the plunger (direct-steam or power-driven) or rotary (centrifugal) type: the latter being driven by a steam turbine or electric motor. Steam-driven pumps are used when there is need of their exhaust for feed-water heating. Centrifugal pumps are generally more compact, and easier to maintain than plunger pumps.

Size.—For plunger pumps, single-acting, if d_w = diameter of water plunger, inches, n = number of water cylinders, S = plunger speed, feet per minute, the discharge in cubic feet per minute is $Snd_w^2 \div 366$, less the leakage and slip. Power-driven pumps are usually single-acting; steam-driven usually double-acting. For double-acting pumps, double the value above. Steam-driven pumps are usually described by giving (1) diameter of steam piston, (2) diameter of water plunger, (3) stroke: thus, 6 by 4 by 6. The steam piston should be somewhat larger in diameter than the water piston so that the pump may function even when worn. Piston speeds are practically never over 100 ft. per minute: half this is a proper allowance. The outside packed-plunger type is more easily kept in good shape, and the pot-valve pump is best where gritty water is used. For hot water, pumps should be brass-fitted.

Centrifugal Pumps.—These contain spirally-bladed impellers which by

revolving accelerate water supplied axially. The velocity is converted into pressure by discharge vanes (turbine pump) or by a spiral casing (volute pump). A single inlet makes for simple construction, especially in multi-stage pumps: a double inlet eliminates end thrust. For high heads, impellers may be connected in series, giving the *multi-stage* pumps. When driven by a steam turbine, speeds must usually be higher than when electrically driven. Designs vary in excellence: an efficiency of 0.70 to 0.80 usually implies fairly high first cost.

Power to Pump Water.—If C = cubic feet per minute, p = pressure (pounds per square inch) including static head, e = pump efficiency; then horsepower = $Cp \div 230e$. Taking boiler horsepower = $120C$, pump horsepower ÷ boiler horsepower = $p \div 27{,}600e$.

Efficiencies.—The leakage loss in a plunger pump may be under 5 per cent when the pump is new, increasing to 30 per cent or more after long wear with gritty water. Wear has little effect on the discharge of a centrifugal pump. Power efficiencies of plunger pumps running at rated capacity vary usually from 0.50 to 0.75 (somewhat less if outside packed), being greater for long-stroke pumps. If the pump varies its discharge by varying its speed, the efficiency holds up well at light loads. If the speed is constant and a trip or unloading valve is used, the efficiency at low discharge rates is very low.[1]

Injectors.—These form inexpensive and useful emergency boiler feeders. They should be supplied with cold water. The efficiency is unimportant, since all heat not employed in pumping or lost in radiation goes back to the boiler. Exhaust steam may be used to operate an injector feeding against boiler pressure. The discharge temperature will usually be around 160°. The weight of steam discharged through the injector is about $Fp \div 70$ pounds per second, where F = minimum nozzle area, square inches and p = steam supply pressure, pounds per square inch, absolute. Under usual conditions, the weight of water delivered is about 10 times the weight of steam.

Feed-water Heaters.—Types include the *open heater* (Cochrane, Webster) in which the exhaust steam and water mix, and the closed heater, in which the water circulates in tubes surrounded by exhaust steam. Open heaters give slightly higher feed temperatures at the same back pressure, or slightly less back pressure at the same feed temperature. Filters, separators, etc., are provided to remove oil from the exhaust. The open heater forms a convenient receptacle for various drips, for the automatic introduction of any cold water make-up supply and for certain forms of feed-water treatment and purification. It may be of the "thoroughfare" type in which all exhaust steam in the pipe passes through the heater, or of the "draw in" type in which a branch from the auxiliary exhaust leads to the heater as a dead end. Open heaters must be located on the suction side of the feed pump and above (preferably 3 ft. or more above) the level of the feed-pump suction valves.

Closed heaters include the coil type with single or multiple coils, the plain straight tube type, the corrugated straight tube and the double tube with water passing through the annular space. Plain tubes must be so arranged with the tube sheets as to permit of expansion.

Operation of Heater.—In a closed heater:

[1] The subject of "Pumps" is covered much more exhaustively in Section III, "The Transportation of Liquids."

$$W_S(Q - h_0) = W_W(h_2 - h_1) = Skt_M,$$

Where, W_S = Weight of steam condensed, pounds per hour,
W_W = Weight of water heated, pounds per hour,
Q = Heat content of entering steam, B.t.u. per pound = $h + xL$ where, h = heat of liquid and L = heat of vaporization, both corresponding with the pressure of the steam, and x = dryness of steam entering the heater,
h_0 = Heat content of condensed steam corresponding with its temperature t_0, B.t.u. per pound,
h_2, h_1 = Heat content of water, B.t.u. per pound, corresponding with its outlet and inlet temperatures t_2 and t_1,
S = Transmitting surface of heater, square feet,
k = Transmission factor, and
t_M = $(t + t_0 - t_1 - t_2) \div 2$ = arithmetical mean temperature difference, where t = temperature corresponding with the pressure of the steam

The value of k is about the same as in surface condenser practice under like conditions.

In an *open* heater, the third member of the equations is disregarded and $t_0 = t_2$, $h_0 = h_2$. The size of such a heater is based by the manufacturer on sufficiently low velocities of steam and water. Ratings will be found liberal.

Economizer.—The economizer is a feed-water heater using boiler flue gases as the heating agent. It usually consists of a single unit for an entire row of boilers, with a gas bypass to the stack. The surface is made up of vertical cast-iron tubes, 4 in. in diameter and 10 or 12 ft. long, through which the water rises. The outsides of the tubes are cleaned of soot by power-operated scrapers. Economizers obstruct the draft and often necessitate the use of fans.

The surface and effect are expressed by

$$0.24W_G(T'_2 - T'_1) = W_W(h_2 - h_1) = SKt_M,$$

in which W_G = weight of gases, pounds per hour,
T'_2, T'_1 = Inlet and outlet temperatures of gases, and $t_m = (T'_2 + T'_1 - T_2 - T_1)/2$. The value of K is around 2.0; that of W_G is related to that of W_W: since with given fuel and given boilers there is per pound of coal a normal weight of feed water and a normal weight of flue gas. The value of W_G/W_S may be as low as 1.7 with the best operation and good coal; it may increase to three or four times this with bad operation. If C = pounds coal burned per hour, the proportion of (gross) saving due to the economizer is $0.24\ W_G(T'_2 - T'_1)/C$. In common practice, $T'_1 - T_2 > 100$. The value of T_1 should exceed 90° to prevent sweating of tubes.

Superheater.—This may be placed in the boiler setting, or separately fired. The Heine type (in the setting) has deflecting dampers to vary the gas flow. Other setting types should be placed in a zone where the gas temperature is at least 1,000°. Forcing the boiler decreases the amount of superheat. Provision should be made for flooding the superheater when it is not in use. It should be designed for at least 150°F. of superheat. Steam-pipe velocities may then be high. While increasing thermal efficiency of engines or turbines, superheat is injurious to metals and often interferes with lubrication. Plain, annular or corrugated (extension surface) tubes may be used. The design equation is

$$W_S(Q' - Q) = 0.24W_G(T'_2 - T'_1) = Skt_m,$$

Where, Q' = Heat content of superheated steam at temperature t'_1, B.t.u. per pound,

Q = Heat content of saturated steam, B.t.u. per pound, and

$t_m = (T'_2 + T'_1 - t' - t)/2$.

In ordinary practice, for setting-type superheaters, k = 2 to 4: for independently fired, k = 8 to 12. If set as prescribed, $(T'_2 + T'_1)$ will be not far from twice $(t' + t)$. The quantity $Q' - Q$ is approximately equal to 0.5 $(t' - t)$. The ratio of W_G to W_S is fixed as in economizers.

Condensers.—From a thermal standpoint, condensers may be classified as mixing (horizontal independent jet, injector or barometric) and non-mixing (surface) types. The equations above given for feed-water heaters will apply: as an error on the safe side, x may be taken as 1.0. The value of t_1 is determined by the available supply of cooling water: in this latitude under summer conditions, it is rarely below 70°. That of t_0 will be in surface condensers 5 to 10° below t, and that of t_2 the same amount below t_0. The value of t_0 determines the best possible vacuum. Orrok gives $k = 350r^2\sqrt{V}$, for clean copper tubes, where V = water velocity in tubes, feet per second, r = ratio of partial pressure of steam to total pressure in the condenser. If condensers and vacuum pumps are tight, r = 0.95.

Small tubes therefore favor high transmission if the water is clean. The length of a tube should be not over 30 times its diameter, and the aggregate tube length (all passes) probably not over 50 times. A multiplicity of passes may be used, the steam striking water passages several times. If s = pitch of tubes on tube sheets (usually about 50 per cent more than the tube diameter) the number of tubes per square foot of tube sheet area is $166 \div s^2$.

The surface condenser is generally used with turbines, both because it usually permits of a better vacuum and because the condensed steam, free from oil, is thereby unmixed with the (usually inferior) circulating water. In engine plants, if circulating water is salt or otherwise unsuitable for boiler feed, the condensed steam may be used for boiler feed if a surface condenser is used, but care must be taken that cylinder oil does not reach the boilers.

Condenser Auxiliaries.—Unless water is available under pressure, a circulating pump is required. The head against which it pumps is that due to difference of levels plus friction head. The quantity of water may be computed from the equation referred to: usually 20 to 40 times the weight of steam condensed for engines: 50 to 100 times, for turbines. Circulating pumps may be of either the plunger or centrifugal type, the latter being particularly adapted for this application (see under boiler feed pumps as to sizes and efficiencies).

The vacuum pump has heretofore been generally of the plunger type. Its displacement should be about 40 times that of condensed steam to be handled, to allow for air and vapor. In the horizontal independent jet condenser, the vacuum and circulating functions are combined in one cylinder. In barometric and ejector condensers, the water velocity (or the head of a column of tail water) will maintain a sufficiently good vacuum for most engines without the use of a vacuum pump. For turbine plants and high vacuum work, three pumps are generally used: one for circulating water, one for pumping out the condensed

steam and one for maintaining the vacuum (dry air pump). Condensers with tail pipes must be placed 35 to 40 ft. above the tail-water level.

Centrifugal air pumps, lately introduced, use the centrifugal action of water to entrain the air. They are simple in operation, though of low efficiency from a power standpoint. The Parsons Vacuum Augmenter uses a steam jet to draw out the air which is then passed over the coils of a small auxiliary condenser. The condensed steam is pumped out in the usual way, but by a pump which need not maintain the high vacuum existing in the condenser, and may therefore be small.

The power required by condenser auxiliaries may vary from 1 to 10 per cent of the main engine output. It rarely pays to attempt better vacuums than 28 and 26 in. for turbines and engines respectively.

Cooling towers and cooling ponds are used where there is a scant supply of circulating water. The vacuum is usually poor and the cost of obtaining it relatively high.

Draft Apparatus.—The draft of a chimney varies directly as its height above the grates. The volume of gas it discharges varies directly as the area, and as the square root of the height. Very low and very high chimneys are both undesirable. For sizes required, see the accompanying tables.

The table gives the horsepower of chimneys computed by Kent method:

Diameter, inches	Height								Effective area, square feet	Actual area, square feet
	60 ft.	80 ft.	100 ft.	125 ft.	150 ft.	200 ft.	250 ft.	300 ft.		
18	25	29	30	36	40	46	51	56	0.97	1.77
24	54	62	69	78	85	98	110	120	2.08	3.14
30	92	107	119	133	146	169	189	206	3.58	4.91
36	141	163	182	204	223	258	288	315	5.47	7.07
42	200	231	258	289	316	365	408	447	7.76	9.62
48	269	311	348	389	426	492	549	602	10.44	12.57
60	437	505	565	632	692	800	894	979	16.98	19.64
72	646	747	835	934	1,023	1,181	1,320	1,447	25.08	28.27
84	896	1,035	1,157	1,294	1,418	1,637	1,830	2,005	34.76	38.48
96	1,186	1,370	1,532	1,713	1,876	2,167	2,423	2,654	46.01	50.27
108	1,517	1,751	1,959	2,054	2,392	2,770	3,098	3,393	58.83	63.62
120	.....	2,180	2,438	2,557	2,986	3,448	3,855	4,223	73.22	78.54
132	.....	2,656	2,970	3,114	3,637	4,200	4,696	5,144	89.18	95.03
144	.....	.....	3,554	3,726	4,352	5,027	5,618	6,155	106.72	113.10
168	.....	.....	4,878	5,115	5,974	6,899	7,713	8,449	146.50	153.94
192	.....	.....	.....	6,724	7,852	9,068	10,138	11,105	192.56	201.06
216	.....	.....	.....	.....	9,987	11,532	12,894	14,123	244.90	254.47
240	.....	.....	.....	.....	12,378	14,293	15,980	17,505	303.53	314.16

For pounds coal burned per hour for any given size of chimney, multiply figures in table by five. Chimneys 25 per cent larger are recommended for low-grade bituminous coal in middle and western states.

Chimney design must be based upon kind of coal required. For overfeed stokers

and hand fires, Fig. 21 (from B. & W. 1913 issue of "Steam") gives the usual draft required for various coals. Artificial draft is generally necessary for steam sizes of anthracite. The following table applies to plants where oil fuel is used.

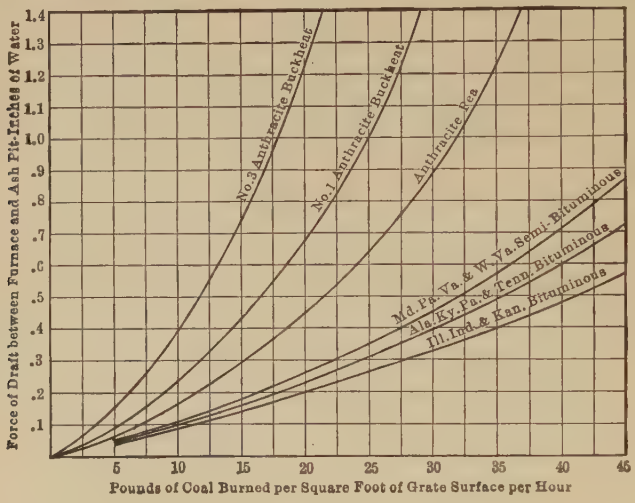


FIG. 21.—Draft for various coals.

STACK SIZES FOR OIL FUEL

Stack diameter inches	Height in feet above boiler-room floor						
	50	70	90	100	120	140	160
18	57	88	103	108	115	120	124
24	110	173	204	214	229	240	249
30	182	288	342	361	388	409	424
36	272	436	521	550	596	629	655
42	382	615	741	785	852	902	940
48	512	830	1,000	1,060	1,160	1,230	1,290
54	660	1,080	1,310	1,390	1,520	1,610	1,700
60	827	1,360	1,660	1,760	1,940	2,060	2,160
66	1,020	1,670	2,050	2,180	2,400	2,560	2,690
72	1,220	2,020	2,490	2,650	2,910	3,130	3,290
84	1,690	2,830	3,490	3,730	4,130	4,430	4,670
96	2,250	3,770	4,670	5,000	5,560	5,980	6,320
108	2,880	4,840	6,030	6,470	7,180	7,770	8,230
120	3,580	6,060	7,560	8,130	9,060	9,800	10,400

Figures represent nominal rated horsepower; table based on centrally located stacks, short direct flues and ordinary operating efficiencies. (Courtesy C. C. Moore & Co., Engrs., and C. R. Weymouth, San Francisco.)

The cheapest and least durable chimney is the guyed steel stack. Self-supporting metal stacks and those of reinforced concrete come next. Radial and hollow brick stacks, properly designed, are cheaper than those of common brick of equal excellence.

Including all types, stacks cost in normal times from $1.50 to $4 per boiler horsepower in large sizes.

Mechanical Draft.—This responds to changing loads and varying atmospheric conditions as the chimney can not, and permits of the use of fuels which could not be burned under any practicable chimney.

Mechanical draft may be afforded by a steam jet, by blast fans (*forced draft*) discharging to the ashpit, or by suction fans (*induced draft*) placed between the boiler and the short stack. Steam jets, whether of the suction or pressure type, are wasteful of steam, though cheap to install. They will not give very high drafts, but tend to prevent clinkering.

Forced draft fans should be used in connection with adequate stacks, so that a blowpipe action on the brickwork may be avoided. Careful regulation is necessary. Induced draft fans must be much larger (the volume of gases handled is greater) and are more costly to maintain because they handle hot and gritty gases.

Size of Fans.—In a given fan in service, the volume delivered varies directly as the speed (revolutions per minute), the pressure (draft) as the square of the speed and the power consumed as the cube of the speed. Hence large fans, though of greater first cost, are cheaper to operate. For similar fans operating at fixed peripheral speed (and therefore fixed draft) the volume and power consumed vary as the square of the diameter. A fan should be selected which at a reasonable revolutions per minute (generally under 275) will give the peripheral speed necessitated by the imposed draft.

FUELS AND FURNACES

Classification of Coals.—Coals obtained west of Pittsburgh are generally of inferior quality. Coal may be classified according to either the *proximate* or the *ultimate* analysis. Thus Frazer's classification:

Kind of Coal	Anthracite	Semi-anthracite	Semi-bituminous	Bituminous
Ratio fixed carbon: volatile:	100 to 12	12 to 8	8 to 5	5 to 0

Also Campbell's classification, based on the ultimate analysis:

Kind of Coal	Anthracite	Semi-Anthracite	Semi-Bituminous	Bituminous	Lignite	Peat	Wood
Ratio C : H:	above 26	23 to 26	20 to 23	11.2 to 20	9.3 to 11.2	below 9.3	7.2

Heat Value.—The heat values of coals are known from calorimetric determinations. The A. S. M. E. formula, based on the ultimate analysis, gives

$$\text{B.t.u. per pound} = 14{,}600\text{C} + 62{,}000\left(\text{H} - \frac{\text{O}}{8}\right) + 4{,}000\text{S},$$

where C, H, O and S are the respective weights of carbon, hydrogen, oxygen and sulphur in 1 lb. of fuel. Where the proximate analysis is given, Lucke proposes

$$\text{B.t.u. per pound} = 14{,}544F + 27{,}000V\left[1 - \frac{1}{\frac{F}{V} + 0.5}\right],$$

Where F and V denote the weights of fixed carbon and volatile in 1 lb. of fuel.

Soft coals have various local classifications, such as the several sizes of "nut" and "screenings." In general, such coals are graded as "lump," "run-of-mine" and "slack," the latter either being the fine, weathered product resulting from exposure of the others, or the finer part of the "run-of-mine."

Typical Coals Etc., Illustrating Classifications

Grade	Name of fuel	C/H	F. C./Vol.	Proximate				Ultimate						B.t.u. per pound	
				H_2O	Vol.	F. C.	Ash	H_2	C	N_2	O_2	S	Ash	Experimental	Calculated
Anthracite	Pennsylvania, Treverton	52.31	12.82	0.84	6.67	85.66	6.83	1.73	90.66	....	0.78	....	6.83	14,025	14,235
Anthracite	Pennsylvania	43.35	32.33	3.45	2.72	87.92	5.90	2.00	86.46	....	2.20	....	5.9	13,471	13,787
Semi-anthracite	Creusot, France	24.40	8.57	1.80	10.10	86.65	1.45	3.66	89.39	....	3.70	....	1.45	15,127	15,220
Semi-bituminous	Pocahontas, run-of-mine	20.27	4.03†	0.80	18.30	73.65	7.25	4.13	83.75	0.85	2.65	0.57	7.25	14,512	14,707
Semi-bituminous	Pocahontas, run-of-mine	20.11	4.04†	0.63	18.62	75.12	5.63	4.25	85.46	0.85	3.24	0.57	5.63	14,733	15,029
Bituminous	Pittsburgh (Carnegie, Pa.)	14.93	1.46	1.07	37.79	55.06	6.08	5.13	76.57	1.64	8.82	1.76	6.08	13,977	14,317
Bituminous	Wakeford, Ohio	14.93	1.43	1.55	37.29	53.34	7.82	4.98	74.39	1.40	6.42	3.44	7.82	13,426	13,982
Bituminous	Marion, Ill., run-of-mine	13.70	1.72	5.96	30.29	52.16	11.59	4.92	67.30	1.43	12.99	1.77	11.59	12,103	12,850
Bituminous	Henryetta, Okla. lump and slack	13.60	1.40	3.87	35.73	50.05	10.35	5.14	69.85	1.29	11.38	1.99	10.35	12,620	13,355
Bituminous	Altoona, Iowa, lump	12.30	0.952	4.52	40.96	38.99	15.53	4.93	60.62	0.93	11.16	6.83	15.53	11,356	12,079
Lignite	Boulder, Col., black	10.60	1.16	13.49	37.11	43.03	6.37	5.75	61.13	1.22	24.95	0.58	6.37	10,791	11,401
Lignite	Sheridan, Wyoming, black	9.60	1.04	17.69	37.96	39.56	4.79	6.09	58.41	1.09	29.99	0.63	4.79	10,355	12,212
Peat	Bohemian*	9.60	0.45	....	68.93	31.07	....	5.96	57.21		36.82		.....	10,625	
Wood	Oak*	8.57	.....	....	.....	.....	....	5.88	50.44		43.69		.....	8,440	
Wood	Norway pine*	8.48	.....	....	.....	.....	....	6.02	51.08		42.90		.....	8,690	

* Dry, ash-free.

† Note disagreement with classification.

STEAM SIZES OF ANTHRACITE COAL

Name	Square mesh screen		Average per cent of Ash
	Pass through, inches	Pass over, inches	
Pea	¾	½	15
Buckwheat No. 1	½	¼	19
Rice	¼	⅛	25
Barley	⅛	1/16	?

Burning Hard Coal.—Anthracite coals require large grate areas and ample draft (page 32). In general, the cost per B.t.u. is less for coals of inferior quality: hence these coals are apt to be cheapest to use if they can be burned with efficiency. Economy in the use of fine sizes of hard coal is chiefly limited by the high ash content, which increases labor, makes it difficult to maintain a good condition of fuel bed, and increases the loss through the grate bars to the ashpit. These objections are partly offset by mixing the coal with a proportion of soft coal—run-of-mine or slack.

Lignites are difficult to burn because of the incombustible nature of much of their volatile content. Extension furnaces for gradual drying of the coal are an aid.

Soft Coal.—The presence of visible smoke from combustion is an indication of loss, but not necessarily of large losses. Other wastes may occur when soft coal is burned: the volatile hydrocarbons not being completely consumed. The following chart is convenient for estimating the density of smoke from chimneys, both as a check on the completeness of combustion and as evidence in case certain chimneys are attacked as nuisances by owners of property near metallurgical plants. (Use this chart at arm's length. The original is a chart 3 × 24 in., supposed to be posted about 50 ft. away.) Smoke darker than the third from the left, emitted for more than 10 per cent of the total time, has been held objectionable.

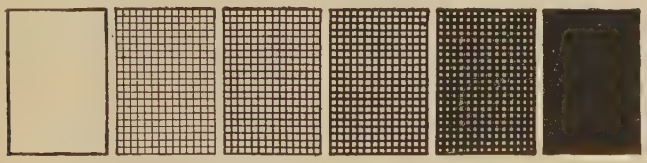


FIG. 22.—Ringelmann's smoke chart.

For the thorough utilization of the hydrocarbons in soft coal, there must be provided:

(*a*) Room for the flame before it strikes the relatively cold heating surface of the boiler. The space and distance necessary vary with the amount and nature of volatile content, and other factors. As much as 12 ft. may be needed.

(*b*) Maintenance of a steady high temperature *at some point* or points of the fuel bed or in the furnace.

(*c*) The supply of an adequate amount of air, thoroughly distributed, at the point where the combustion of the hydrocarbons is to take place.

(*d*) The supply of fuel, and as far as possible, the manipulation of the fire, without opening of fire doors.

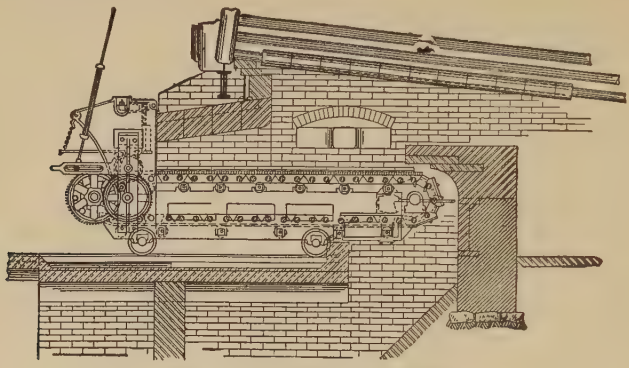


FIG. 23.—Chain-grate stoker.

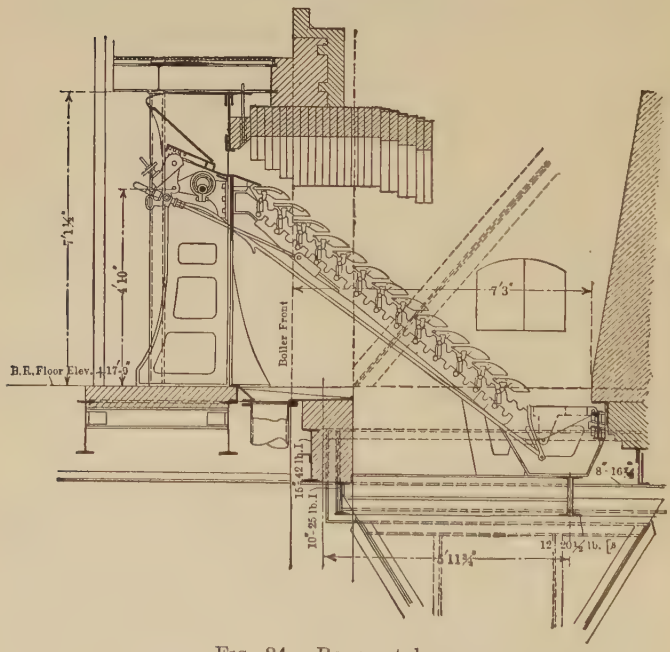


FIG. 24.—Roney stoker.

These principles underlie the operation of smokeless furnaces and mechanical stokers. Stokers vary greatly in maintenance cost and in the associated cost of maintaining brickwork. In large plants, they save labor. A modern form of the old "Dutch oven" extension furnace, will give little smoke even when hand fired, if the furnace is not driven hard. The flames are kept hot, because they do not strike the boiler surface until combustion is complete.

In the *chain grate stoker* (see illustration) the coal is carried forward continuously on a moving chain, the ashes being dropped at the back end. The gases from the fresh fuel pass over the hotter coke fire on the back part of the grate. In *underfeed*

stokers (American, Jones) the fuel is delivered by a worm or ram to the under side of the fire, and the rising smoke passes through the incandescent fuel. *Inclined grate stokers* like the Roney (inclined aft) and Murphy (inclined toward the middle, transversely) use a coking plate at the top and an arch overhead, giving a deep furnace, the gas current being more or less downward from the coking plate over the coke fire on the grate.

Stokers require power for operation and if used with coarse or varying sizes of coal will often require the use of a crusher, which consumes an appreciable amount of power. Overhead coal storage, which permits of gravity feed to the stoker hoppers, also greatly increases the boiler-house cost. *Coal feeders* (for hard coal) are mainly in the experimental stage. They will occasionally save labor cost and possibly slightly improve efficiency by removing the necessity for opening fire doors. *Steam jets* are sometimes useful in diluting smoke.

Air Supply.—Considering coal as pure carbon, with pure dry air as the agent for supplying oxygen, the maximum temperature is attained when 11.57 lb. of air are supplied per pound of carbon. If less than this weight of air is supplied, CO will be formed instead of CO_2, and the heat evolution per pound of carbon burned will be 4,450 B.t.u. instead of 14,500 B.t.u., for each pound burned to CO instead of CO_2. If more than 11.57 lb. of air are supplied, the reactions may be perfect, but the furnace temperature will be reduced. Moreover, since the heat lost to the stack is proportional to the weight of flue gas, this loss of heat increases with excess air supply. A close approximation to the ideal air supply is desirable: the degree of approximation is indicated by the per cent of CO_2 in the flue gases which is a maximum when the air supply is exactly right.

In practice, some excess of air is found to be necessary, and with amounts too close to the ideal, CO and free O_2 may be found present simultaneously. This is to be avoided by thorough distribution of the entering air, avoidance of leaks through furnace and setting walls, and forced draft coupled with rather inadequate suction so as to approximate atmospheric pressure in the furnace (balanced draft). Coals containing hydrogen or hydrocarbons require more air than does pure carbon. Oil requires less air than coal.

Coal Handling and Storage.—The link-belt conveyor is the handling device in most general use, in spite of its high maintenance cost. Traveling cranes with traveling hoists running off on stationary runways, mast and gaff outfits with industrial railways and (for ashes) the pneumatic system may sometimes be considered. In all cases, quick unloading is particularly desirable. This implies a trestle for rail deliveries and a grab-bucket for boats where the water is sufficiently deep to permit the use of large boats. In large plants, ashes should accumulate in a bin over the team driveway, so that carts may be quickly loaded. Coal storage for winter requirements is now generally necessary. There is both shrinkage and deterioration in outside storage, particularly with soft coal. Danger of spontaneous ignition should be met by using ventilating boxes and by having steam and (if possible) some excavating device for extinguishing or digging out the fire when it does occur. Coal weighs 50 to 60 lb. per cubic foot in its usual commercial forms.

Fuel Oil.—North American fuel oils show a rather steadily progressive increase in density and heat value and decrease in volatile yield as we advance from the East toward the West and Southwest. Eastern oils are paraffin base:

Western oils usually asphaltum base or a mixture of the two bases. The latter are difficult to use in internal combustion engines, but make good boiler fuels. With fair approximation,

$$\text{B.t.u. per pound oil} = 18{,}650 + 40(B - 10)$$

Where, B = Baumé hydrometer reading. If s = specific gravity, $B = (140 \div s) - 130$. High boiler efficiencies are easily possible with fuel oil. The principal impurities to be avoided are moisture and sulphur. The latter, in some oils, is as high as 3 per cent.

Commercial products are derived from crude petroleums by fractional distillation at increasing temperatures:

Gasoline, 56°Bé. and upward;
Kerosene, 44 to 49°Bé.;
Gas oil, solar oil, 35°Bé.;
Lubricants and residuum, 28°Bé. and downward.

Fuel oil may be either a crude (from which it may be necessary to remove sulphur, silt, moisture and gasoline), a distillate (gas oil or solar oil) or a residuum. The heat

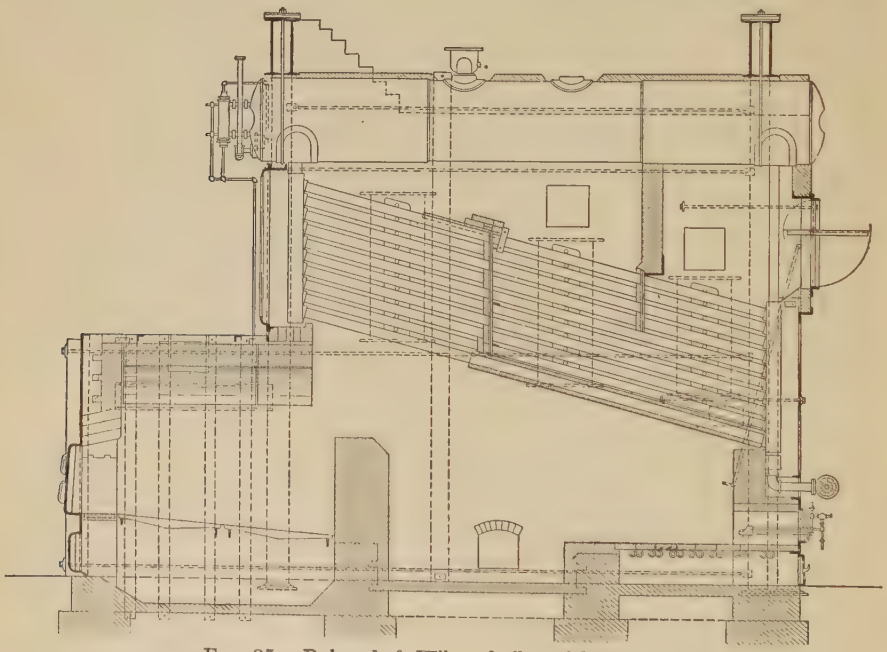


Fig. 25.—Babcock & Wilcox boiler with Dutch oven.

value and also the ease of utilization depend on the density. In steam boiler work, any American oil down to 14°Bé. may be employed. The oils most available for oil engines are the "solar" and gas types. These are expensive. Mexican oil down to 18°Bé. has been successfully used in high compression (300 lb.) engines, but the cost of heating the oil by live stream is considerable. The possibility of using a new oil in an engine can be positively determined only by actual trial.

Besides the main storage (in large plants not in thickly settled sections) underground storage tanks are usually installed close to the boiler room. Burners should never be fed by gravity. The oil should be heated (usually to about 180°) before

delivery to burners. Assuming a minimum tank temperature of 30°, with exhaust steam as the heating agent, the transmitting surface necessary (which may be wrought iron pipe) is about 1 sq. in. per pound of oil per hour if the oil velocity in the tube is to be kept down to the point where the pumping head is reasonable. The weight of steam will be about one twentieth the weight of oil.

Burners are usually operated by steam or compressed air, the former involving simpler and cheaper equipment but possibly a somewhat greater operating cost, since up to 4 per cent of all the steam generated will be expended in atomizing the oil. This is high-pressure steam, and the necessary make-up feed water must be supplied. A common pressure for oil fed to steam-operated burners is 30 lb. gage, maintained by a pressure regulator on the pump. Ample straining should be provided with provision for cleaning strainers without interrupting operation. Burners should be made so as to be readily cleaned, with provision for easy renewal of tips. A single burner will discharge enough oil for a 400-hp. boiler. Automatic regulation of steam pressure and oil delivery in proportion to the load is desirable.

Oil permits of a closely regulated air supply and high furnace temperature. The requirements for good combustion are similar to those for highly volatile soft coals (p. 35). It is important that the intense heat be well distributed and to avoid the blowpipe effect due to impingement from the burner on metal or brickwork. This often leads to the use of two burners, where one would have sufficient capacity. It also explains the furnace design illustrated. This has the "back shoot" burner, the air being admitted through checkerwork. Peepholes through the furnace walls are desirable.

The fire risk is minimized by isolating the main storage tanks, providing a smothering steam supply to the inside of all tanks, and avoiding overheating of the fuel. *Gas tar* is burned like oil. It is particularly heavy and has a low flash-point, so that the temperature must be carefully adjusted. Its heat value is around 14,000 B.t.u. per pound.

MISCELLANEOUS SOLID FUELS

	Composition, per cent			B.t.u. per pound
	Carbon	Ash	Moisture	
Peat	35, dry	3 to 36	6 to 20, air-dried	6,000 to 10,000, dry
Wood	50, dry	Usually < 1	18 to 60	8,000 to 9,000, dry
Straw	36, dry	5	16	5,000 to 6,500, wet
Shelled corn	..	..	..	7,800 to 8,500, dry
Charcoal	84	3	12	12,850
Bagasse	43 to 47	1½ to 3	40 to 54	8,000, dry
Tan bark	52, dry	1.4, dry	66	9,500, dry

Wood comes next to cellulose in the progressive series from that compound to anthracite. It is available as cord wood, slabs, edgings and refuse, the last being often "hogged" to shreds. A cord of hemlock weighs 1,200 lb.; of hickory, 4,500 lb. The soft woods are the higher in heat value. Woods contain about 6 per cent of hydrogen and under 1 per cent of nitrogen. Rates of combustion for refuse should be high (150 lb.) and a strong natural draft used. Extension furnaces are essential for driving off moisture. Large furnace space, a thick fuel bed and an undisturbed fire are recommended. Refuse should be chuted to

the furnace, one chute supplying 30 sq. ft. of grate. Heating surfaces must be frequently cleaned.

Bagasse may now be burned without auxiliary fuel, on a hearth leading to brick-walled passages. About 0.3 in. draft will give a rate of combustion of 250 to 300 lb. **Tan bark** also requires a high rate of combustion. 1 lb. of ground bark yields slightly over 2 lb. of spent tan. **Black ash liquor** is available in soda-process pulp mills. It is burned in rotary extension furnaces with auxiliary fuel: high concentration of liquor decreases the consumption of the latter. The ash is reclaimed and the steam produced is a by-product.

Gas Fuels.—The important gas fuels are listed below, with typical analyses as given by Lucke:

Name	Analysis by volume per cent								Density, pounds per cubic foot, 32°F.	B.t.u. per pound (low value)
	CH_4	H_2	C_6H_6	CO	O_2	C_2H_4	CO_2	N_2		
Natural gas	98.4	.....	0.1	0.25	0.25	...	.....	.....	0.0446	21,301
Coal gas, coke oven gas	34.3	42.00	2.0	6.00	1.10	2.0	2.50	10.10	0.0403	14,384
Blast-furnace gas	0.2	2.74	...	28.60	....	...	11.39	57.06	0.0812	1,323
Water gas	4.41	45.57	...	44.85	0.50	0.1	4.45	0.12	0.0456	7,228
Producer gas	0.2	15.00	...	26.10	0.20	...	5.30	53.2	0.0696	1,934

Coal gas is the product of slow distillation of coal with exclusion of air. Blast-furnace gas results from the combustion of coal in a restricted supply of air: its ideal composition would be, by weight, CO, 34.4; N, 65.6. Water gas is made by the action of steam on incandescent coal, and should consist ideally of H and CO only: the process is necessarily intermittent. Producer gas is a combination of blast-furnace gas and water gas, the supply of steam being so limited as to permit of a continuous process. In all of the manufactured gases, the hydrocarbons (CH_4, C_6H_6, C_2H_4) come from the coal either directly or as the result of a breakdown of other hydrocarbons. The hydrogen in coal gas and blast-furnace gas has a similar origin. The CO and CO_2 in coal gas are due to a partial (and objectionable) combustion of the carbon in the coal.

For steam boilers, fuels are natural gas and blast furnace gas. A large number of small burners (about 30 hp. each) is used for the former. Burners are thoroughly distributed. As with fuel oil, they may be directed either slightly downward toward the rear or may face from the bridge wall toward the front. The combustion space should be large. The air supply is drawn in around the gas jet. High efficiency is easily possible. For *blast furnace* gas, use a furnace space of 2 cu. ft. per rated horsepower, 0.8 sq. in. of gas passage area per horsepower and 2 in. blast (based on 6 to 8 in. pressure in main). Provision must be made for removal of dust. With all gas burners, the gas and air supply should be separately controllable.

Powdered Coal.—The fuel is blown into the furnace so as to burn in suspension. Furnace construction is much the same as for oil, and combustion is very perfect. Dryers are employed to drive off moisture before grinding. On account of its lower moisture content, run-of-mine coal may be preferred to slack, in spite of its higher price. The chief cost is for fixed charges in the grinding and feeding equipment, and for power. From 50 cts. to $1 per ton seems to be

the usual range of costs for preparation. This may be compared with the losses incident to gasifying the coal in a producer plant, due regard being paid to the following factors:

1. The costs of coals suitable for gasifying and pulverizing may differ.
2. Pulverizing equipment and gasifying equipment involve different initial costs.
3. Gas produced from coal may be used either as a boiler fuel or direct, in the cylinder of a gas engine.

Briquetting involves a cost of $1 to $2 per ton over that of raw fuel.

Wood.—Solid wood fuel prepared for the purpose, is occasionally used under boilers in the Northwest. Split fir at $3 per cord, delivered, is about equivalent economically to the local low-grade coals at $2.20 per ton. Refuse wood fuels (slab, edgings, chips and sawdust) are more frequently employed. Suitable methods of feeding must be provided: fine particles may be blown long distances. If the moisture content is large, a special furnace on the Dutch oven principle may be found necessary. Woods contain very little ash.

STEAM-PLANT PIPING

Materials.—Pipe may be wrought (welded soft steel), cast-iron, brass, galvanized wrought, galvanized sheet riveted. *Wrought pipe* is standard ("full") weight, extra strong or double extra strong. Inside sizes are nominal up to 13 in., above which size pipe is purchased by the actual external diameter and the standard thickness is $\frac{3}{8}$ in. The heavier pipes are of the same external diameter and threading. *Cast-iron pipe* is made with either flanged or bell and spigot joints, the latter being used underground with lead and oakum calking. Classes *A* to *H* (A. W. W. Association, U. S. Cast Iron Pipe & Foundry Co.) describe various weights of the latter, increasing from those suitable for 100 ft. head to weights recommended for 800 ft. head. Common "soil pipe" should not be used. Standard lengths are 12 ft.: sizes from 3 to 84 in. Flanged end pipe is obtainable in the same classifications and weights, sizes ranging from 3 to 48 in. *Brass pipe* should be specified as "iron pipe size." It will then match cast-iron fittings and threads, and will be sufficiently rugged. *Galvanized wrought pipe* is used in the smaller sizes only. Galvanized sheet riveted pipe may be employed for low pressures and large diameters. The spirally-riveted form is the stronger.

Joints.—Common screwed joints may be used up to $2\frac{1}{2}$ in., with ground joint unions for the smaller and flange unions for the larger sizes. Screwed joints are more readily made tight but add to the difficulty of alteration or replacement. "Making up" by machine is usually desirable for sizes above 2 in. Flanged joints offer much variety. The commonest form runs the flange up on the thread while the pipe end is still back of the flange face. The *tongued and grooved* flange decreases the area of flange bearing and therefore increases the unit contact pressure due to bolt tension. This joint may if desired be formed by the pipe ends themselves. Riveted and peened joints are rightfully obsolete. The best plain flange joint is made by running the pipe clean through the flange (without shouldering on the thread) and facing off on centers. In the various modifications of the *Van Stone* joint, a very soft pipe is used, this is slipped through the flange without threading and the pipe end is flanged over and faced off so that there is no possibility of a thread leak. *Welded* flanges are

Standard Full-weight Wrought-iron and Steel Pipe[1]

National Tube Co.

Diameter			Nominal thickness, inches	Circumference		Transverse areas			Length of pipe per square foot of		Length of pipe containing 1 cu. ft., feet	Nominal weight per foot, pounds	Number of threads per inch of screw
Nominal internal, inches	Actual external, inches	Approximate internal diameter, inches		External, inches	Internal, inches	External, square inches	Internal, square inches	Metal, square inches	External surface, feet	Internal surface, feet			
⅛	0.405	0.27	0.068	1.27	0.85	0.13	0.06	0.07	9.44	14.15	2,513.00	0.24	27
¼	0.540	0.36	0.088	1.70	1.14	0.23	0.10	0.12	7.08	10.49	1,383.30	0.42	18
⅜	0.675	0.49	0.091	2.12	1.55	0.36	0.19	0.17	5.66	7.76	751.20	0.57	18
½	0.840	0.62	0.109	2.63	1.95	0.55	0.30	0.25	4.55	6.15	472.40	0.85	14
¾	1.050	0.82	0.113	3.30	2.59	0.87	0.53	0.33	3.64	4.64	270.00	1.13	14
1	1.315	1.05	0.134	4.13	3.29	1.36	0.86	0.50	2.90	3.65	166.90	1.68	11½
1¼	1.660	1.38	0.140	5.22	4.34	2.16	1.50	0.67	2.30	2.77	96.25	2.27	11½
1½	1.900	1.61	0.145	5.97	5.06	2.84	2.04	0.80	2.01	2.37	70.66	2.72	11½
2	2.375	2.07	0.154	7.46	6.49	4.43	3.36	1.07	1.61	1.85	42.91	3.65	11½
2½	2.875	2.47	0.204	9.03	7.75	6.49	4.78	1.71	1.33	1.55	30.10	5.79	8
3	3.500	3.07	0.217	11.00	9.63	9.62	7.39	2.24	1.09	1.25	19.50	7.57	8
3½	4.000	3.55	0.226	12.57	11.15	12.57	9.89	2.68	0.96	1.08	14.57	9.11	8
4	4.500	4.03	0.237	14.14	12.65	15.90	12.73	3.18	0.85	0.95	11.31	10.79	8
4½	5.000	4.51	2.246	15.71	14.16	19.64	15.96	3.68	0.76	0.85	9.02	12.54	8
5	5.563	5.05	0.259	17.48	15.85	24.31	19.99	4.32	0.69	0.76	7.20	14.62	8
6	6.625	6.07	0.280	20.81	19.05	34.47	28.89	5.59	0.58	0.63	4.98	18.97	8
7	7.625	7.02	0.301	23.96	22.06	45.66	38.74	6.92	0.50	0.54	3.72	23.54	8
8	8.625	8.07	0.276	27.10	25.35	58.43	51.15	7.28	0.44	0.47	2.82	24.69	8
8	8.625	7.98	0.322	27.10	25.07	58.43	50.02	8.41	0.44	0.48	2.88	28.55	8
9	9.625	8.94	0.344	30.24	28.08	72.76	62.72	10.04	0.40	0.43	2.29	33.91	8
10	10.750	10.19	0.278	33.77	32.01	90.76	81.55	9.21	0.36	0.37	1.76	31.20	8
10	10.750	10.14	0.306	33.77	31.86	90.76	80.75	10.01	0.36	0.38	1.78	34.24	8
10	10.750	10.02	0.366	33.77	31.47	90.76	78.82	11.94	0.36	0.38	1.82	40.48	8
11	11.750	11.00	0.375	36.91	34.56	108.43	95.03	13.40	0.33	0.35	1.51	45.56	8
12	12.750	12.09	0.328	40.06	37.98	127.68	114.80	12.88	0.30	0.32	1.25	43.77	8
12	12.750	12.00	0.375	40.06	37.70	127.68	113.10	14.59	0.30	0.32	1.27	49.56	8
13	14.000	13.250	0.375	43.96	41.60	153.86	137.81	16.05	0.27	0.29	1.04	54.57	8
14	15.000	14.250	0.375	47.10	44.70	176.62	159.39	17.23	0.25	0.27	0.90	58.57	8
15	16.000	15.250	0.375	54.24	47.90	200.96	182.55	18.41	0.24	0.25	0.75	62.58	8

[1] Black steel pipe in random lengths with threads and couplings shipped unless otherwise specified.

TEMPLATES FOR DRILLING STANDARD AND LOW-PRESSURE FLANGED FITTINGS[1]
American Standard

Size, inches	Diameter of flanges, inches	Thickness of flanges, inches	Bolt circle diameter, inches	Number of bolts	Size of bolts, inches
4	9	15/16	7½	8	⅝
4½	9¼	15/16	7¾	8	¾
5	10	15/16	8½	8	¾
6	11	1	9½	8	¾
7	12½	1 1/16	10¾	8	¾
8	13½	1⅛	11¾	8	¾
9	15	1⅛	13¼	12	¾
10	16	1 3/16	14¼	12	⅞
12	19	1¼	17	12	⅞
14	21	1⅜	18¾	12	1
15	22¼	1⅜	20	16	1
16	23½	1 7/16	21¼	16	1
18	25	1 9/16	22¾	16	1⅛
20	27½	1 11/16	25	20	1⅛
22	29½	1 13/16	27¼	20	1¼
24	32	1⅞	29½	20	1¼
26	34¼	2	31¾	24	1¼
28	36½	2 1/16	34	28	1¼
30	38¾	2⅛	36	28	1⅜
32	41¾	2¼	38½	28	1½
34	43¾	2 5/16	40½	32	1½
36	46	2⅜	42¾	32	1½
38	48¾	2⅜	45¼	32	1⅝
40	50¾	2½	47¼	36	1⅝
42	53	2⅝	49½	36	1⅝
44	55¼	2⅝	51¾	40	1⅝
46	57¼	2 11/16	53¾	40	1⅝
48	59½	2¾	56	44	1⅝

[1] These templates are in multiples of four, so that fittings may be made to face in any quarter and bolt holes straddle the center line. Bolt holes are drilled ⅛ in. larger than nominal diam. of bolts.

used with satisfactory results. Flanges should always be ring-faced or spot-faced on the back, and of suitable thickness to permit of this process. *Gaskets* are of rubber, rubber with wire-insertion, rubber with canvas insertion, asbestos with wire insertion, corrugated copper and corrugated steel.

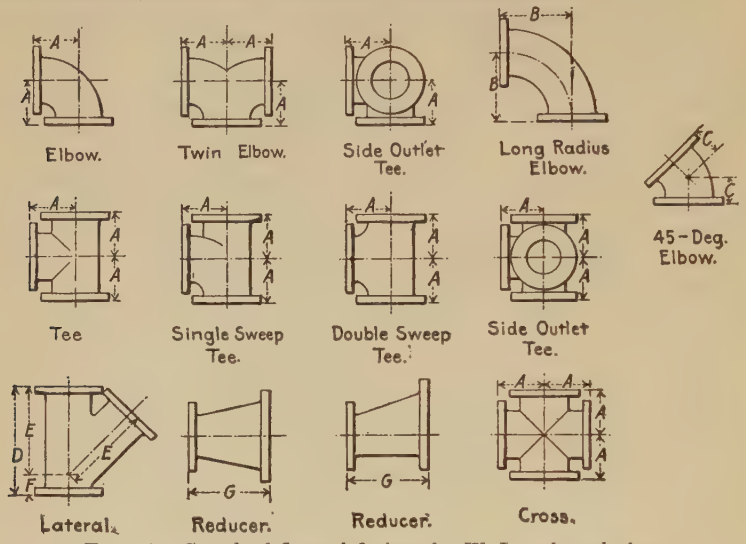


FIG. 26.—Standard flanged fittings for W. I. and steel pipe.

TEMPLATES FOR DRILLING EXTRA HEAVY FLANGED VALVES AND FITTINGS[1]
American Standard

Size, inches	Diameter of flanges, inches	Thickness of flanges, inches	Bolt circle diameter, inches	Number of bolts	Size of bolts, inches
3	8¼	1⅛	6⅝	8	¾
3½	9	1 3/16	7¼	8	¾
4	10	1¼	7⅞	8	¾
4½	10½	1 5/16	8½	8	¾
5	11	1⅜	9¼	8	¾
6	12½	1 7/16	10⅝	12	¾
7	14	1½	11⅞	12	⅞
8	15	1⅝	13	12	⅞
9	16¼	1¾	14	12	1
10	17½	1⅞	15¼	16	1
12	20½	2	17¾	16	1⅛
14	23	2⅛	20¼	20	1⅛
15	24½	2 3/16	21½	20	1¼
16	25½	2¼	22½	20	1¼

[1] These templates are in multiples of four, so that fittings may be made to face in any quarter and bolt holes straddle the center line. Bolt holes are drilled ⅛ in. larger than nominal diam. of bolts.

Fittings.—These are made both standard and extra heavy, and flanged fittings should follow standard flange drilling with holes straddling centers. Bell and spigot fittings ("specials") are available in weights to suit those of the pipe. Galvanized and brass fittings are generally with screwed ends. Malleable fittings are rarely used.

GENERAL DIMENSIONS OF STANDARD FLANGED FITTINGS—STRAIGHT SIDE
American Standard

(All dimensions in inches. Letters refer to sketches in Fig. 26)

For sizes from 32 in. to 100 in., dimension G = size of pipe (S) and A-A = $2A$; also, in lower half of table, d and t = diameter and thickness of flanges, and t_1 = minimum metal thickness of body.

Size S	Face to face, tees and crosses A-A	Center to face, ells, tees and crosses A	Center to face, long-radius ells B	Center to face, 45-deg. ells C	Face to face, laterals D	Center to face, laterals E	Center to face, laterals F	Face to face, reducers G	Diam. of flanges	Thickness of flanges	Minimum metal thickness of body
4	13	6½	9	4	15	12	3	7	9	15/16	½
4½	14	7	9½	4	15½	12½	3	7½	9¼	15/16	½
5	15	7½	10¼	4½	17	13½	3½	8	10	15/16	½
6	16	8	11½	5	18	14½	3½	9	11	1	9/16
7	17	8½	12¾	5½	20½	16½	4	10	12½	1 1/16	5/8
8	18	9	14	5½	22	17½	4½	11	13½	1⅛	5/8
9	20	10	15¼	6	24	19½	4½	11½	15	1⅛	1 1/16
10	22	11	16½	6½	25½	20½	5	12	16	1 3/16	¾
12	24	12	19	7½	30	24½	5½	14	19	1¼	1 3/16
14	28	14	21½	7½	33	27	6	16	21	1⅜	⅞
15	29	14½	22¾	8	34½	28½	6	17	22¼	1⅜	⅞
16	30	15	24	8	36½	30	6½	18	23½	1 7/16	1
18	33	16½	26½	8½	39	32	7	19	25	1 9/16	1 1/16
20	36	18	29	9½	43	35	8	20	27½	1 11/16	1⅛
22	40	20	31½	10	46	37½	8½	22	29½	1 13/16	1 3/16
24	44	22	34	11	49½	40½	9	24	32	1⅞	1¼
26	46	23	36½	13	53	44	9	26	34¼	2	1 5/16
28	48	24	39	14	56	46½	9½	28	36½	2 1/16	1⅜
30	50	25	41½	15	59	49	10	30	38¾	2⅛	1 7/16

S	A	B	C	d	t	t_1
32	26	44	16	41¾	2¼	1½
34	27	46½	17	43¾	2 5/16	1 9/16
36	28	49	18	46	2⅜	1⅝
38	29	51½	19	48¾	2⅜	1 11/16
40	30	54	20	50¾	2½	1¾
42	31	56½	21	53	2⅝	1 13/16
44	32	59	22	55¼	2⅝	1⅞
46	33	61½	23	57¼	2 11/16	1 15/16
48	34	64	24	59½	2¾	2

(From MARKS' *Mechanical Engineers' Handbook*)

General Dimensions of Extra Heavy Flanged Fittings—Straight Side
(All Dimensions in Inches. Letters refer to sketches in Fig. 26)

Size	Face to face, tees and crosses AA	Center to face, ells, tees and crosses A	Center to face, long-radius ells B	Center to face, 45-deg. ells C	Face to face, laterals D	Center to face, laterals E	Center to face, laterals F	Face to face, reducers G	Diam. of flanges	Thickness of flanges	Minimum metal thickness of body
3	12	6	7¾	3½	14	11	3	6	8¼	1⅛	9/16
3½	13	6½	8½	4	15½	12½	3	6½	9	1 3/16	9/16
4	14	7	9	4½	16½	13½	3	7	10	1¼	5/8
4½	15	7½	9½	4½	18	14½	3½	7½	10½	1 5/16	5/8
5	16	8	10¼	5	18½	15	3½	8	11	1⅜	1 1/16
6	17	8½	11½	5½	21½	17½	4	9	12½	1 7/16	¾
7	18	9	12¾	6	23½	19	4½	10	14	1½	1 3/16
8	20	10	14	6	25½	20½	5	11	15	1⅝	1 3/16
9	21	10½	15¼	6½	27½	22½	5	11½	16¼	1¾	⅞
10	23	11½	16½	7	29½	24	5½	12	17½	1⅞	1 5/16
12	26	13	19	8	33½	27½	6	14	20½	2	1
14	30	15	21½	8½	37½	31	6½	16	23	2⅛	1⅛
15	31	15½	22¾	9	39½	33	6½	17	24½	2 3/16	1 3/16
16	33	16½	24	9½	42	34½	7½	18	25½	2¼	1¼

Weights of Cast-iron Flanged Fittings for Steam
(American Standard Dimensions)

Size, inches	Approximate weight per piece, pounds							
	Standard (125 lb.)				Extra heavy (250 lb.)			
	Ell	45°Ell	Tee	Cross	Ell	45°Ell	Tee	Cross
3	30	27	45	58	46	38	70	90
3½	37	33	55	74	57	44	75	115
4	45	38	67	89	67	61	100	140
4½	46	43	75	100	85	70	120	170
5	63	53	90	121	95	85	130	190
6	75	68	115	152	125	105	190	250
7	100	90	150	200	160	145	235	325
8	120	100	170	236	190	175	280	370
9	150	130	220	305	240	195	330	480
10	205	160	285	400	320	250	450	580
12	285	230	430	570	450	380	680	900
14	390	300	550	750	640	520	970	1,300
15	440	330	660	800	750	570	1,050	1,400
16	525	400	760	1,000	840	675	1,255	1,675

(From Marks' *Mechanical Engineers' Handbook*)

Valves.—Valves are of the globe or gate type (the angle valve may be of either, being merely a modified body form). The gate valve gives a straight passage: the globe valve is better for use as a regulator, but should have its stem always in a horizontal position. High-pressure valves larger than 6 in.

Service	Size	Pipe	Fittings and valves	Joints
1. High-pressure steam (above 160 lb.) with superheat.	Small	Extra strong, wrought	Cast steel, extra heavy: valves special, as below	Screwed
	Large	Standard or extra strong, wrought	Cast steel, extra heavy: bronze valve spindles: special valve necks: non-corrosive seats, rings and plug faces	Van Stone or welded flanges, corrugated steel gaskets
2. High-pressure steam, saturated	Small	Wrought iron, standard	Cast iron, extra heavy	Screwed
	Large	Wrought iron, to suit joints	Cast iron, extra heavy	Van Stone or welded flanges, corrugated copper gaskets.
3. Steam pressure from 30 to 80 lb., saturated.	Small	Wrought iron, standard	Cast iron, standard	Screwed
	Large	Wrought iron, standard	Cast iron, standard	Faced flanges, rubber gaskets with wire insertion.
4. Exhaust steam.....	Small	Wrought iron, standard	Cast iron, standard	Screwed
	Large	Cast iron (light) or galvanized sheet	Cast iron, standard	Plain-faced flanges: rubber wire-insertion gaskets
5. Cold or warm water, underground.	Small	Cast iron or galvanized standard	Cast iron or galvanized, standard	Cast iron, plain flanges, galvanized screwed.
	Large	Cast iron, brass and steel	Cast iron, brass and steel (valves special for salt water)	Bell and spigot, calked with lead and oakum
6. Coldwater, above ground.....	Small	Galvanized or black wrought, standard	Galvanized or cast iron, standard	Screwed
	Large	Cast iron, flange	Cast iron, flange, usually standard	Plain-faced flanges, rubber gaskets with wire insertion
7. Hot water.....	Small	Brass, standard	Brass or cast iron, standard or extra heavy	Screwed
	Large	Brass, standard, or cast iron	Cast iron	Faced flanges after making up: rubber gaskets with wire insertion

should have bypasses. Small valves (especially those of brass) have threaded tops: larger valves have bolted tops. Valves are obtainable in "standard," "extra heavy" and some special weights.

Size of Pipes.—Velocities in pipes were formerly 6,000 ft. per minute for live and 4,000 ft. for exhaust steam. With superheated steam velocities up to 15,000 ft. are regularly used. Very large pipes should be avoided. The size should be calculated for the allowable pressure drop:

$$w = 87 \sqrt{\frac{p\, D\, d_1^5}{L\left(1 + \frac{3.6}{d_1}\right)}}$$

Where, w = Weight of steam carried, pounds per minute,
p = Pressure drop from beginning to end of line, pounds per square inch,
D = Weight of steam per cubic foot at the mean pressure (see steam table, p. 49),
d_1 = Actual internal diameter of pipe, inches, and
L = Length of pipe, feet.

For each entrance, elbow or valve, there is additional resistance to flow about equivalent to that set up by a straight pipe length of about 60 diameters.

Heat Loss.—Bare pipe emits about 3 B.t.u. per hour per square foot for each degree difference of temperatures of steam in the pipe and air surrounding it. Drafts of air greatly increase this. Commercial coverings reduce this loss by 70 to 90 per cent. In general all piping containing heat which is still to be used is worth covering. It usually pays to cover flanges, valve bodies, etc., on high pressure steam lines: but the covering should be applied only after tightness has been demonstrated and should be so applied as to interfere as little as may be with repairs. Sectional covering (jacketed or banded) is used in preference to plastic where possible. Durability of covering should be considered as well as insulating properties and cost.

Outdoor steam pipes may be designed for low pressure drop and low heat loss. They should never be buried in the ground. For large installations, tunnels give good access and minimum loss, but are very expensive ($20 upward per running foot). Overhead lines carried on poles or towers should have waterproofing outside the insulation.

Specialties.—The **back-pressure valve** is a low-pressure safety valve usually applied to vent surplus steam from a low-pressure exhaust or heating system. The **automatic relief valve** is of similar purpose but more elaborate design, applied on vacuum lines to give an emergency vent should the vacuum fail. Pressure regulating or **reducing valves** are used to maintain a steady low pressure for heating or process work less than the prevailing high pressure of the plant. When used on heating systems, the low pressure for which they are set should be less than that at which the back-pressure valve operates. *Foot valves* are used at the base of vertical suction pipes to retain some water when the pump is shut down. A *strainer* should be incorporated here. Boiler *blowoff* valves should be installed in pairs and should be rugged, with renewable seats. *Automatic stop* valves, placed next to boiler nozzles, stop the flow of steam should a pipe burst. *Non-return* valves similarly stop the return flow in case of rupture of a boiler tube. *Check valves* are of the swing (flapper) and globe (vertical spindle) types. They are used on the boiler-feed line. *Steam traps* drain off water

without permitting the escape of steam as such. Both bucket traps and expansion traps are available. *Steam separators* remove moisture from the current of steam. *Oil separators* or *grease extractors* remove moisture and oil. They should be used on exhaust lines from reciprocating engines and pumps to heating systems, etc. All separators require traps.

Piping Design.—*Expansion* must be controlled and allowed for. It may be as much as 3 in. per 100 ft. of pipe length. Expansion joints are to be avoided. They are available in both the packed stuffing box and corrugated types. The best method (though limited in application) of providing for expansion is by *pipe bends*. These also decrease the heat emitting surface, reduce the number

TABLE OF PROPERTIES OF SATURATED STEAM[1]

Pressure in pounds per square inch	Temperature, Fahrenheit	Total heat in heat units above 32°F.		Heat of vaporization of latent heat (L) in heat units $L=H-h$	Density or weight in pounds of 1 cu. ft.	Volume in cubic feet of 1 lb.	Factor of equivalent evaporation at 212°F.
		In the steam (H)	In the water (h)				
1	101.99	1113.1	70.0	1043.0	0.00299	334.5	0.9661
2	126.27	1120.5	94.4	1026.1	0.00576	173.6	0.9738
3	141.62	1125.1	109.8	1015.3	0.00844	118.5	0.9786
4	153.09	1128.6	121.4	1007.2	0.01107	90.33	0.9822
5	162.34	1131.5	130.7	1000.8	0.01366	73.21	0.9852
6	170.14	1133.8	138.6	995.2	0.01622	61.65	0.9876
7	176.90	1135.9	145.4	990.5	0.01874	53.39	0.9897
8	182.92	1137.7	151.5	986.2	0.02125	47.06	0.9916
9	188.33	1139.4	156.9	982.5	0.02374	42.12	0.9934
10	193.25	1140.9	161.9	979.0	0.02621	38.15	0.9949
15	213.03	1146.9	181.8	965.1	0.03826	26.14	1.0003
20	227.95	1151.5	196.9	954.6	0.05023	19.91	1.0051
25	240.04	1155.1	209.1	946.0	0.06199	16.13	1.0099
30	250.27	1158.3	219.4	938.9	0.07360	13.59	1.0129
35	259.19	1161.0	228.4	932.6	0.08508	11.75	1.0157
40	267.13	1163.4	236.4	927.0	0.09644	10.37	1.0182
45	274.29	1165.6	243.6	922.0	0.1077	9.285	1.0205
50	280.85	1167.6	250.2	917.4	0.1188	8.418	1.0225
55	286.89	1169.4	256.3	913.1	0.1299	7.698	1.0245
60	292.51	1171.2	261.9	909.3	0.1409	7.097	1.0263
65	297.77	1172.7	267.2	905.5	0.1519	6.583	1.0280
70	302.71	1174.3	272.2	902.1	0.1628	6.143	1.0295
75	307.38	1175.7	276.9	898.8	0.1736	5.760	1.0309
80	311.80	1177.0	281.4	895.6	0.1843	5.426	1.0323
85	216.02	1178.3	285.8	892.5	0.1951	5.126	1.0337
90	320.04	1179.6	290.0	889.6	0.2058	4.859	1.0350
95	323.89	1180.7	294.0	886.7	0.2165	4.619	1.0362
100	327.58	1181.9	297.9	884.0	0.2271	4.403	1.0374
105	331.13	1182.9	301.6	881.3	0.2378	4.205	1.0385
110	334.56	1184.0	305.2	878.8	0.2484	4.026	1.0396
115	337.86	1185.0	308.7	876.3	0.2589	3.862	1.0406
120	341.05	1186.0	312.0	874.0	0.2695	3.711	1.0416
125	344.13	1186.9	315.2	871.7	0.2800	3.571	1.0426
130	347.12	1187.8	318.4	869.4	0.2904	3.444	1.0435
140	352.85	1189.5	324.4	865.1	0.3113	3.212	1.0453
150	358.26	1191.2	330.0	861.2	0.3321	3.011	1.0470
160	363.40	1192.8	335.4	857.4	0.3530	2.833	1.0486
170	368.29	1194.3	340.5	853.8	0.3737	2.676	1.0502
180	372.97	1195.7	345.4	850.3	0.3945	2.535	1.0517
190	377.44	1197.1	350.1	847.0	0.4153	2.408	1.0531
200	381.73	1198.4	354.6	843.8	0.4359	2.294	1.0545
225	391.79	1201.4	365.1	836.3	0.4876	2.051	1.0576
250	400.99	1204.2	374.7	829.5	0.5393	1.854	1.0605
275	409.50	1206.8	383.6	823.2	0.5913	1.691	1.0632
300	417.42	1209.3	391.9	817.4	0.644	1.553	1.0657
325	424.82	1211.5	399.6	811.9	0.696	1.437	1.0680
350	431.90	1213.7	406.9	806.8	0.748	1.337	1.0703
375	438.40	1215.7	414.2	801.5	0.800	1.250	1.0724
400	445.15	1217.7	421.4	796.3	0.853	1.172	1.0745
500	466.57	1224.2	444.3	779.9	1.065	0.939	1.0812

[1] KENT, "Mechanical Engineer's Pocket-Book," New York, 1913, p. 836.

of joints and decrease the cost of installation. They should be used liberally. Ample use of special fittings, where joints may thereby be saved, is also good practice. *Drainage* is of prime importance. Pipes approximately horizontal should slope downward in the direction of steam flow and should be drained at low points. Branches should be taken from tops or sides of pipes supplying them. High-level high-pressure drips may be returned directly to the boiler. Clean drips from other points may enter an open feed heater or hot well. Greasy low-pressure drips are usually wasted.

ELECTRICAL MACHINERY

Units and Definitions.—Units of voltage, current, resistance and power are the volt (V), ampere (I), ohm (r) and watt (P). For *direct current*, $I = V/r$, $P = VI$. 1 hp. = 746 watts. 1 kilowatt (kw.) = 1.34 hp. = 1,000 watts. The *heating effect* is $P_h = rI^2$. In *alternating currents*, the voltage and current are out of phase because of inductance, and both vary continuously during each revolution of the generator. If the variation may be expressed as a sine curve against time (which is approximated in usual practice), $P = VI \cos \theta$ where θ is the angular phase difference and $\cos \theta$ is the *power factor*. The *effective* voltage or current is the **maximum** $\div \sqrt{2}$. An alternating-current generator may send out a series of alternating currents of different *phase* over different conductors. The angle between these different phases remains constant. The so-called single-phase system is a two-phase arrangement with one terminal of each winding connected to a common junction point (neutral point) while the other terminals of the windings feed the live wires. A similar three-phase winding constitutes the Y arrangement: whereas the Δ arrangement connects all windings in series with line wires brought out from the junctions. If I_l, V_l denote line current and voltage, n = number of windings, the following relations hold:

	Δ	Y
Phase current	$I_l/2 \sin \frac{\pi}{n}$	I_l
Phase voltage	$V_p = V_l$	$V_p = V_l/2 \sin \frac{\pi}{n}$
Voltage to neutral	$V_p/2 \sin \frac{\pi}{n}$	V_p
Volt-amperes	$nV_lI_l/2 \sin \frac{\pi}{n}$	$nV_lI_l/2 \sin \frac{\pi}{n}$

A *period* (alternating current) is the time required for an alternating current or voltage to complete a cycle of values. *Frequency* = periods per second.

Series arrangements are end-to-end: current passing from one device to another in succession. *Multiple* or *parallel* arrangement exists where devices (lamps, for example) are connected across mains, each lamp being independent of all others.

The *circular mil* is the area of a circle of $\frac{1}{1000}$ in. diameter. *Torque*, T, is tangential pull at 1 ft. radius, in pounds. If R = revolutions per second, the corresponding horsepower is $\pi TR/550$.

Systems.—Direct current is used for short distances of transmission and low voltages, or where speed variation is necessary. In most industrial applications direct current gives higher power efficiency but reduced ruggedness and reliability.

Enclosed machines are more fully protected from dust and dampness, but are poorly ventilated, must be larger to avoid excessive heating and cost more.

Motors.—The *shunt motor* is used for constant speeds, and has high efficiency over a narrow range of speeds and a wide range of loads. Full-load efficiencies vary with size: say from 0.70 to 0.90 for sizes from ½ to 30 hp. Speeds are high for small powers. At a given power, high speed implies lower weight and cost (*speed* times *weight* is approximately constant). The cost per pound for motors of 25 hp. is only about half that of 1 hp. motors. The weight per horsepower is around 100 lb. for all sizes. This is the type of motor used for driving shafting, machine tools, fans, reciprocating pumps and other machines where load variations and starting conditions are not severe and a steady speed is required.

The *series motor* is used where frequent starts under load are necessary as for traction, and hoisting. Speeds increase as loads (torques) decrease and the motor may "run away" if the load is entirely removed (as by breakage). Power output remains about constant over a wide range of speeds: efficiency is constant for a wide speed range but a narrow power range. Full-load efficiencies vary in about the same way as those for shunt motors, but the maximum is about 3 per cent lower. Five hundred-volt series motors weigh 30 to 60 lb. per horsepower. Costs per pound are about constant for sizes from 50 to 200 hp.

The *compound motor* is a compromise type used where frequent starts are necessary and some variation in speed permissible. It has both shunt and series windings, which may act together (cumulative compounding) or oppositely (differential compounding).

Voltages commonly used are 110 to 125 for lighting and small motors, or 220 to 300, for power and lighting; and 500 to 600 for power, particularly traction (for alternating-current voltages see below). The current required by a direct-current motor = (hp. × 746) ÷ (*efficiency* × *voltage*). Torques of motors otherwise similar in design are proportional to *number of poles times current*.

Constant speed direct-current motors are usually started by decreasing the amount of an external resistance in the armature circuit. The starting current may exceed full-load current by about 50 per cent. From 15 to 30 sec. should be allowed to bring a motor up to speed. This may be insured by automatic operation. The rheostat may incorporate low-voltage and overload releases, and should contain the motor-field circuit closing mechanism. Ordinary rheostats (starters) cost from 3 to 6 per cent as much as motors, the smaller sizes costing relatively more.

Speed Variation.—Speed control of direct-current motors is effected by:

(*a*) Multiple voltage system (three-wire, etc.) Several generators and wires are used, affording a choice of voltages for the armature circuit and therefore of speeds. (Voltage is proportional to r.p.m.) The field voltage is kept constant. Shunt motors are to be used if stability of speed is desired. The efficiency remains good, but the motor is operating at low power output when its speed is low, and the cost of investment is high.

(*b*) Rheostatic control of armature circuit. This varies voltage and speed, but the speed at any rheostat position varies with the load and the efficiency is low.

(*c*) Rheostatic control of field circuit. This increases speed by weakening the field. Commutating (inter) poles are desirable if the speed range is to be great. Results are about as with method (*a*).

To reverse the direction of rotation of a direct-current motor, reverse the terminals to *either* the field or the armature winding: not to both.

Group vs. Individual Drives.—Individual motor drives are vastly more convenient, but greatly increase installation cost. The effect on every day

efficiency is variable. While the group drive involves losses due to friction of shaft bearings, the individual motor is apt to operate at a poor load factor and therefore at low efficiency. Where one machine is likely to be operated alone frequently, it may well have its own motor. The group drive does not permit of electrical methods of speed variation. Individual motors require a more careful determination of power requirement.

Generators.—Electric generators are shunt or compound wound. The latter can be made to give more uniform voltage. To operate compound wound generators together (feeding the same wires) an equalizer is used. This makes a common connection to all machines at a point between the armature and the series field and divides the series field currents in proper proportions between the machines. Generators to be operated together must have similar electrical characteristics. Voltage is proportional to revolutions per minute $\times$ *number of poles* $\times$ *magnetic flux per pole.* Generators (and motors) are rated as to capacity with reference to the rise of temperature in a stated length of time. In common practice a machine will not heat more than 90°F. above a surrounding air at 77°F. when running continuously at rated load. *Regulation* refers to variation of speed or voltage over a range of loads (usually no load to full load) and is expressed as a percentage of the full-load value. Full-load *efficiencies* of direct-current generators are from 0.80 to 0.93 for sizes from 1 to 500 hp. The variation of efficiency with load is about the same as for shunt motors. Weights are almost directly proportional to output. Costs per pound decrease somewhat as the size increases. High-speed (revolutions per minute) leads to low cost per kilowatt of capacity, but the variation in cost is less rapid than the change in speed.

Alternating-current Generators.—These may be of *synchronous* or *induction* type (some should always be of the former). Most large or high-voltage induction alternators are built with the fields rather than the armature revolving. Alternators may be single-phase, two or three-phase. The first are 30 per cent heavier and more costly and less efficient than the others: three-phase current is most economical of copper for transmission. Generator voltages range up to 13,000. *Frequency = revolutions per second* $\times$ *number of poles* $\div$ 2. Frequencies of 25 and 60 cycles are standard. The latter is desirable for lighting or where static transformers are used. The former leads to reduced inductive voltage drop and is therefore better for long transmissions: it adds considerably to generator cost. It is to be preferred for frequent overloads and high starting torques in motor applications, or where synchronous motors or rotary transformers are to be driven. Alternators are *rated* (at 100 per cent power factor) on the basis of temperature rise: for example, 90° may be the limit at rated load, starting from 77°; or 99° for a 25 per cent overload lasting 2 hr. and immediately succeeding a long run at rated load. Polyphase alternators have full-load *efficiencies* from 91 to 97 per cent, increasing with size. Weights increase less rapidly than outputs: costs per unit of capacity decrease as capacity increases. Large units run at the lower speeds. For a given size, high speed means low cost. Turbine-driven machines operate at particularly high speeds and are so compact that special provision must be made for ventilation.

For the operation of alternators in parallel, the supply of power from the respective prime movers must be determined by governors having the same speed-load character-

istics, the generators must have similar starting characteristics and their excitations must be adjusted to equalize power factors.

Alternating-current Motors.—Motor voltages are commonly 110, 220 or 440. There are three general types of motor: commutator, synchronous and induction. *Commutator* (single phase) motors are used for low voltages on railway and hoisting work. They cost 50 to 100 per cent more than direct-current motors. Their characteristics are like those of direct-current series motors: the efficiency is fairly high, but the power factor is low at low speeds. *Synchronous* polyphase motors give the highest efficiencies and power factors, are inexpensive and give steady speeds: but will not start under load: if single-phase, they will not start of themselves at all, and the efficiency is low.[1] If heavily overloaded, they will stop. Direct current is used for field excitation. Voltages up to 13,000 may be used directly, in synchronous motors with revolving fields.

Induction motors are most commonly used, the stator being the primary. They are rated like generators, on the basis of temperature rise. They are essentially constant-speed machines. The *squirrel-cage* type requires a large starting current, and is started by fractional voltage taps. The *wound-rotor* type is preferred, generally, for applications where starts are frequent or severe. Sizes up to 200 hp. may have starting resistance mounted inside the rotor: large sizes have external resistance and the rotor is then provided with collector rings and brushes.[2]

Single-phase and two-phase induction motors usually have independent phase windings. Three-phase motors are connected in Y or Δ. In Y, the current per phase equals the line current and the voltage per phase is line voltage divided by $\sqrt{3}$: in Δ, the voltage per phase equals line voltage and the current per phase is line current divided by $\sqrt{3}$. In either case, *current* per phase is proportional to horsepower divided by (*efficiency* $\times$ *voltage per phase* $\times$ *power factor*). Polyphase induction motors have full-load efficiencies of 0.80 to 0.90 in sizes from 1 to 200 hp., 25-cycle. Corresponding power factors are 0.78 to 0.92. Sixty-cycle motors give efficiencies very slightly lower and full-load power factors about the same. The power factor decreases at light loads, especially for single-phase machines: to obtain high values implies high first cost of motor. Maximum efficiency and power factor are usually realized at about three-fourths rated load. The breakdown point (maximum safe load) should be at least 50 per cent in excess of the rating, and no load imposed should equal this.

Weights and costs vary with power and speed about as do those of direct-current motors. Large machines run at slow speeds. Voltage has little influence on cost. *Starters* cost two to three times as much as those of direct-current motors.

There are no generally applicable methods of speed variation for alternating-current motors. All methods involve a considerable sacrifice of efficiency.

Transformers.—Static transformers are ordinarily used to transform one alternating-current voltage to another, and are rated on the basis of temperature rise. The efficiency may be as high as 0.97 at full load. The curves show variation in efficiency and secondary voltage with load. The energy lost appears as heat and in large sizes is removed by surrounding the coils with oil, by pumping oil or water through or by an air blast. A separate transformer may be used on each phase of a polyphase system, or one three-phase unit may replace three single-phase transformers. The former arrangement may give more spare units: the latter decreases cost and increases efficiency. Costs increase slightly with primary voltage and vary inversely with frequency.

[1] For starting, an auxiliary motor of another type may be used, or the synchronous motor may have a squirrel-cage winding in the pole face, so that it starts as an induction motor.

[2] Single-phase motors may be started (*a*) by incorporating a commutator, (*b*) by hand and (*c*) by operating them as polyphase motors, a "phase splitter" being used to throw two or more circuits out of phase with each other.

The *rotary transformer* is used to change alternating to direct current when both are of about the same voltage. One set of windings answers for the alternating-current motor and the direct-current generator. Full-load efficiency, about 0.93. Where the voltages differ greatly, the more expensive *motor generator* set must be used (efficiency about 0.85). The two machines then have only the shaft and bearings in common. Such a unit may be used to change frequency if desired: parallel operation then presents some difficulties. A *rotary condenser* is a synchronous motor introduced at some point where there is an easily started load to improve the power factor of a system largely made up of induction motors.

Storage batteries are used for direct-current stand-by service, improvement of regulation, excitation of alternators, etc. Cells are in series and in sufficient number to give the required voltage. Charging is by direct current. Ventilation is necessary for removal of gases from the battery room. There are two types: the *alkaline* (Edison) using nickel hydroxide and iron plates in a potassium hydroxide electrolyte: and the *lead*, with plates of lead and lead peroxide in dilute sulphuric acid. Capacities vary, and are expressed in ampere-hours, based on an assumed time of discharge (8 hr. for lead, 5 hr. for alkaline). Efficiencies at such normal rate are around 0.70. During charging, the voltage is increased (1.7 to 1.95 for alkaline, 2 to 2.8 for lead, per cell). During discharge, the voltage falls: 1.54 to 0.88 for alkaline, 2.0 to 1.75 for lead. Costs are very high: in normal times, from $100 to $200 per kilowatt developed when discharging in 1 hr. This is three or four times the capacity realized at the normal discharge rate.

Heating Processes.—Electric furnaces are of the *arc*, *induction* or *resistance* type. The last are used for warming of rooms, requiring 1 to 2 watts per cubic foot of room contents at zero outdoor temperature. The resistance element must be non-oxidizable if it is exposed to the air. Electric house warming is too costly for any but the most exceptional use. In industrial furnaces, electricity gives a high temperature (up to 4,000°C.), localized and susceptible of close regulation. The energy is employed to heat up the charge, sometimes to melt or vaporize it or to produce chemical reaction, and (wastefully) to cover conduction and flue gas losses and electrical (resistance) losses external to the furnace. Furnace linings must be refractory (carbon), but these should be backed by walls of non-conducting material. *Arc* furnaces are built for capacities up to 3,000 kw. per pair of electrodes (three-phase current) for operation at either atmospheric or higher pressure. *Induction* furnaces are virtually immense static transformers, the low-tension windings of which are replaced by baths of molten metal, for example. The amount of current is limited by the so-called "pinch effect." In *resistance* furnaces, the charge itself may constitute the resistance element.

Electroplating, Etc.—The energy required for electroplating, galvanoplasty, detinning and electrolytic refining of metals varies with the metals involved and solutions used. From 1 to 100 amp. may be needed per square inch of cathode surface, at 0.1 to 4 volts per cell. Direct current is supplied from small generators at 5 or 6 volts, and a separate rheostat is required for each cell or tank.

Distribution of Electricity.—Sizes of wires must be suitable for maximum current from the standpoint of heating effect (see table) and must generally be based on the corresponding drop of voltage. For primary mains and feeders a

Size of wire, B. & S. gage	Maximum allowable current-amperes	
	Rubber insulation	Other insulation
6	50	70
4	70	90
2	90	125
0	125	200
0000	225	325
Circular mils		
500,000	400	600
1,000,000[1]	650	1,000

[1] Used mainly for underground work.

drop of 5 or 10 per cent is allowed: for secondary mains and service wires, 2 to 5 per cent. The diversity factor (sum of maximum demands of the parts of a system, divided by the maximum demand of the whole system), enters into the matter. If I = current in amperes, v = drop per conductor in volts, l = length of one conductor in feet, the actual cross-section of copper necessary, in circular mils, is $A = klI \div v$, where $k = 11$ for direct current. For alternating current, k varies with the power factor, etc., and may be as high as 17 at 0.80 power factor.

Wiring Specifications.—Interior wiring may be:

(*a*) *Open.*—On cleats or insulators: rubber insulated if in damp places.

(*b*) *Moulding.*—For under 300 volts: forbidden in concealed or damp places: no joints or taps in the moulding: rubber insulated: wood moulding forbidden in elevator shafts: metal moulding limited to 1,320 watts and must be completely connected and grounded.

(*c*) *Concealed Knob and Tube.*—Often forbidden, but cheap. Rubber insulation: wires 5 in. apart, 1 in. from adjacent surface, supported every 53 in.

(*d*) *Rigid Conduit.*—Most expensive type. Sizes ½ to 4 in. Grounded. Rubber insulation. Lined or unlined conduit may be obtained, the "lining" consisting of a layer of insulation.

(*e*) *Flexible Conduit.*—Interlocking metallic strips. Sizes $\frac{5}{16}$ to 2 in. Specifications otherwise as for rigid conduit.

(*f*) *Armored Cable.*—Rubber insulated and covered with interlocking metallic strips.

Direct current is generally distributed by two-wire or three-wire constant "potential" systems. The latter (with all three wires of the same size) saves five-eighths the copper necessary for the former. The gross saving is less. Alternating current is distributed as:

Two-phase, Three-wire.—Each single-phase current has its own outgoing wire. The return wire serves both. Voltage between two outside wires is 41 per cent more than that between outside and return wire. Current in return wire 41 per cent greater than in either outside wire. An unbalanced system.

Two-phase, Four-wire.—Two complete single-phase circuits. The neutrals of the two circuits may be connected, in which case the voltage across circuits is 71 per cent of that between the wires of a phase. Extensively used. In the *five-wire* system, the common neutral is extended as a fifth wire.

Three-phase, Three-wire.—Three single-phase circuits, the return currents neutralizing. Balanced when the phases are equally loaded.

Three-phase, Four-wire.—The fourth wire a neutral: lamps connected between any outside wire and a neutral wire. Rarely used: six-wire system also in infrequent use.

Relative Voltages and Relative Weights of Copper (Pender)

System		Relative Voltages between outer wires		Relative weight of copper[1]	
		Neutral grounded	Neutral insulated	Neutral grounded	Neutral insulated
	Direct-current, two-wire..	100	50.0	100	400
Unity power factor	Single-phase, two-wire...	71	35.5	200	800
	Two-phase, four-wire....	71	35.5	200	800
	Three-phase, three-wire..	61	35.5	200	600

[1] Not including neutral wires.

Primary Supply.—In large works, with alternating current locally generated, primary voltages of 2,000 to 2,400 are common. For distribution to substations, 5,000 to 110,000 volts may be used. The losses in transmission are partly constant (corona, core loss of transformers, parts of the losses of all rotating apparatus, meter losses and losses at constant-current devices, if there are any); and partly variable with the load, usually varying as the square of the current carried.

Voltage Regulation.—Loading tends to pull down voltage, especially toward the outer ends of lines. The extreme drop at maximum load is limited as already described. Fluctuations due to bad regulation may introduce further drop, decrease of motor output and efficiency, decrease in life and illuminating power of lamps, etc. The extreme voltage variation on lighting circuits is rarely allowed to exceed 5 per cent: on power circuits it may be 10 per cent and is sometimes more. In three-wire or alternating-current systems, the best voltage condition is obtained when the loads on the circuits are equal (or are those for which the circuits were designed). The *motor-generator* used on a three-wire system takes current from the lightly loaded side and delivers current to the other side. The cost of operation is that due to the losses in the machine and the neutral current need not be taken back to the generator. In alternating-current systems (single-phase three-wire) the *balancing coil* may be used. This consists of two transformer windings connected in series across the outside wires. The neutral is connected from between the two windings. The lightly loaded side acts to excite its winding as a transformer primary, and current is thereby transferred by way of the other winding to the other side of the circuit. In polyphase systems, the phases are of course balanced (so far as power applications are concerned) by using polyphase motors.

Electric Lighting.—The value of lights is measured in *candlepower*. The candlepower of a light varies according to the direction from which it is viewed. If C = candlepower in a stated direction which makes an angle α with a plane to be illuminated, the distance from the light to this plane being l ft., then illumination in *candle feet* = $E = C \cos \alpha / l^2$. The candle feet required for

various services range from 1.0 upward. The *lumen* or unit of light flux is the quantity of light flowing outward from a source of 1 cp. in the solid angle subtended by $\frac{1}{12.5664}$ the surface of a sphere. If the average candlepower in all directions is I_s (mean spherical candlepower) the lumens flowing are $12.5664I_s$. Lumens required $= EA/F$ where A = area of room or surface, square feet, F = factor of utilization. The value of F depends on the shape of the room, absorptions and reflections, and on the variation in candlepower of the light source with direction. It may be as low as 10 per cent for indirect lighting[1] and up to 65 per cent for bare lamps in lightly tinted rooms.

INCANDESCENT LAMPS

Type	Watts	Lumens	Life, hours
Carbon	20–120	50–419	600–2,000
Gem	20–100	52–422	650–1,000
Mazda (vacuum tungsten)	10–500	70– 5,089	500–1,500
Mazda (gas-filled tungsten)	750–1,000	13,000–19,000	
Series (vacuum), 4 amperes[1]	35–350	318– 3,563	
Series (gas-filled)	57–500	800–10,000	

[1] Also made for 3.5, 5.5, 6.6 and 7.5 amp. in the same sizes.

In general, alternating-current arc lamps are less efficient than direct-current, and 220-volt lamps are to be avoided. Enclosed arcs have longer life between trims and greater steadiness, but the efficiency is less. Flaming arcs have a short interval between trims and must be hung high. Luminous arcs give off brown fumes, may be used only outdoors, and give an unsteady light. *Efficiency* is measured by lumens per watt, or (inversely) by watts per candlepower, usually (as tabulated) the mean candle power of the lower hemisphere.

PURCHASE OF POWER

Water for power is available in some localities. It is usually sold at a fixed price per year for all the water that can flow through an opening of stated dimensions above which an agreed head is to be maintained. The buyer pays whether he uses the water or not. For small installations, the water consumed may be measured by a Venturi meter. The disposition of tail water is sometimes a factor.

Steam (or other) shaft power is furnished as a part of the rental privilege in some cases. In the case of steam the average power demand is estimated by taking indicator cards at the prime mover, with and without the power load in question. Steam as such may be sold by steam meter or (for heating) on the basis of the cubic feet or square feet of space rented.

Electric power is the form commonly sold. The purchaser is apt to overestimate his average load (basing it on his installed capacity) and to underestimate the installation costs incidental to the purchase of power. Annual rates are sometimes made (especially for hydroelectric power) for 10-hr. or

[1] In direct lighting, the lamps are exposed. In indirect lighting, no lamps are visible, their light being reflected from walls or ceilings. In the direct-indirect system, part of the light is directed downward through shades, while the remainder strikes the ceiling and is reflected downward.

ARC LAMPS

Circuit	Type	Current	Amperes	Power factor	Volts	Watts	Mean lower hemisphere candlepower	Life per trim, hours
Series	Carbon	Open direct current	6.6, 7.5, 9.6	....	50	330–480	395–690	18
		Enclosed direct current		....	75	500–720	479+	125
		Enclosed alternating current		0.84	77	425–535	232+ and 291†	80–125
	Flaming carbon	Open direct current	6.6, 7.5, 9.6,	....	55[a] or 110[b]	660[c] and 715+[b]	1,890[c] and 2,058+[b]	17–20
		Enclosed alternating current		0.75	60	450[d]	1,595[d]	100–150
	Metallic or luminous	Open direct current	4.0, 5.5, 6.6	....	75	312–515	535–1,340	125–175
Multiple	Carbon	Enclosed direct current	5.0, 6.5, 3[e]	....	110, 220	550, 715	379–559	100–150
		Enclosed alternating current	6.0, 7.5	0.65	110, 220	430, 540	371, 395	100–125
	Flame carbon	Enclosed direct current	6.5	....	110	715	1,740	100–125
		Enclosed alternating current	7.5	0.62	110	510	1,600	100–125
	Quartz-tube mercury		3.8–2.0	....	100–625	380–1,250	1,000–3,500	

+ 6.6-amp. lamp. † 7.5-amp. lamp. [a] Inclined carbons. [b] Vertical carbons. [c] 12-amp. lamp. [d] 10-amp. lamp.
[e] for 220 volts only.

24-hr. service. The customer pays for an estimated power requirement, which must not be exceeded, and pays whether he uses the power or not. *Meter rates* are usually based on consumption: the greater the consumption, the lower the rate. A fair basis for rate is an annual charge proportional to the maximum demand (not to the connected load) plus a meter rate for current actually used. An equivalent method is to charge one rate for the first 100,000 kw.-hr. in any month, a lower rate for the next 100,000, and so on.

Contract clauses should provide for interruptions of service, which may sometimes be offset by provisions releasing the purchaser from liability in case of strike or fire in his plant. The phase, frequency and voltage of current *at the point of measurement* should be specified. Services furnishing a high power factor should be somewhat more attractive to the vendor.

PRODUCER GAS

Operation of Producer.—The essential feature is a partial combustion of the fuel. Carbon is converted to CO, itself a combustible suitable for use in an engine or furnace. The original heat of combustion of the carbon is thereby partly converted to sensible heat. Since the gas should be cooled before use in an engine, the maximum ideal efficiency with pure air and pure carbon on the "cold gas basis" is only 0.70.

By introducing some steam as well as air, the sensible heat which would otherwise be wasted is employed to decompose steam. It then reappears as heat of combustion of hydrogen, increasing the cold gas efficiency, which is

$$e' = [10{,}220(1 - x) + 6{,}900y] \div (Q_s + Q_a + 14{,}650),$$

where x = Proportion of C burning to CO_2,

y = Weight of steam introduced per pound of C, in lb.

Q_s = Heat furnished by entering steam

Q_a = Heat furnished by entering air

} B.t.u. per pound of C.

The value of e' may reach 0.83 to 0.85 (ideal), from which radiation losses must be deducted. Slight excesses of steam or air always lead to the formation of some CO_2. The "hot gas efficiency" (furnace applications) is similarly

$$e = [10{,}220(1 - x) + 6{,}900y + Q_g] \div (Q_s + Q_a + 14{,}650),$$

where Q_g = heat carried off by the gas as sensible heat, B.t.u. per pound of C.

Use of Steam.—Steam not only increases efficiency: it lowers the fuel bed temperature and thus discourages clinkering and fusion of furnace walls. It reduces sensible heat loss, but if supplied in excess decreases the rate of gasification and (ultimately) the efficiency. When the steam supply is intermittent, water gas is formed during its admission. This is so rich in hydrogen as to be objectionable for use in an engine (causing preignition) and the intermittent process is adapted mainly for furnace work or for combined furnace and engine plants. The proportions of air and steam, in the continuous process, should be kept constant for a given fuel. This is approximately accomplished by various forms of automatic regulation, but charging the fire and sudden changes in load may nevertheless produce considerable variations, reflected in the heat value of the gas. *Exhaust gas* from the engine may take the place of steam, the theory of operation being precisely the same. This method is less widely applied, although it leads to the formation of a gas (free from hydrogen) better adapted for efficient use in the engine.

Types of Producer.—In the suction type, air and steam are drawn through the fuel bed by the suction of the engine, or by an exhaust fan which afterward delivers the gas to the engine under a slight pressure. The former arrangement is used for small plants only. The latter permits of cleaning while running (therefore of continuous operation) and is the most popular type today. Pressure producers use forced draft blowers. Their operation is attended with some discomfort and they must be shut down for thorough cleaning. Combination producers employ both pressure and

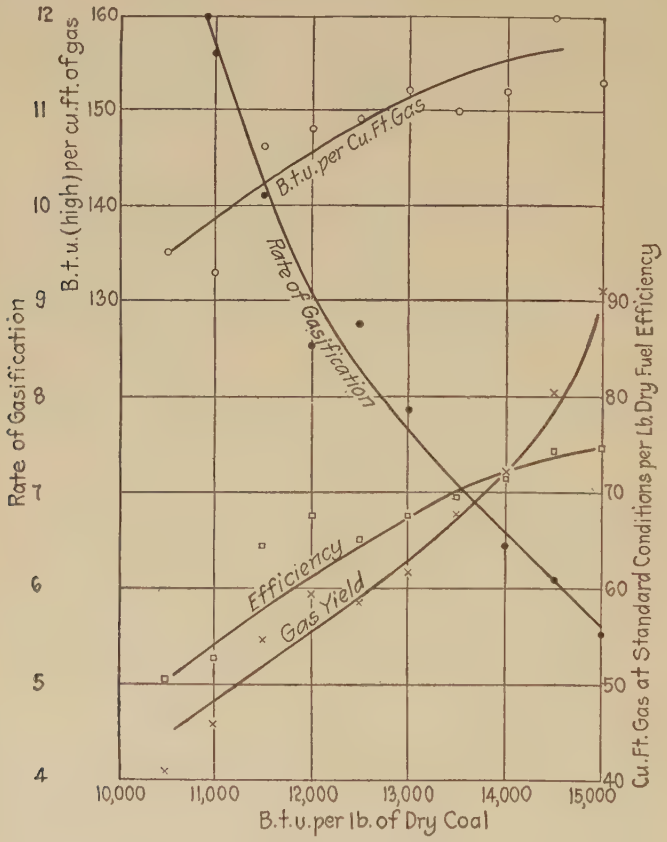


Fig. 27.—Gas producer characteristic curves.

suction, atmospheric pressure being maintained above the fuel bed. Up-draft and down-draft producers are defined by their names. Double-zone producers have down-draft in the upper half, up-draft in the lower half and the gas discharge at mid-height.

Capacity.—Rates of gasification are high when temperatures are high: *i.e.*, when the cold gas efficiency is low. Average rates in suction producers are 5 to 20 lb. per square foot of fuel bed horizontal cross-section area per hour. A mean rate of 10 lb. may be expected for anthracite coal.

Fuels.—Success may be expected with confidence when anthracite (pea or No. 1 buckwheat) is used. The factors of chief importance are moisture, ash, tar, clinkering and heat value. Soft coal is used with success in down-draft or double-zone producers,

which act to "fix" the condensible tars, but trouble may occur from clinkering. Crude oil has been used experimentally, but gas cleaning becomes a serious problem.

Composition of Gas.—Using pure air and pure carbon in the chemically necessary proportions, the gas would consist (by weight) of 0.345 CO and 0.655 N. Blast-furnace gas (which is producer gas incidentally manufactured) approximates this composition more nearly than do any of the producer gases specially made. Using x air and y steam, with symbols as above, gas constituents per pound of carbon are as follows:

	Weight, Pounds	Volume, Cubic Feet
CO	$\frac{7}{3}(1 - x)$	$31.8(1 - x)$
CO_2	$\frac{4}{3}x$	$31.7x$
H_2	$\frac{1}{9}y$	$21y$
N_2	$3.35\ [\frac{4}{3}(1 + x) - \frac{8}{9}y]$	$60.6(1 + x) - 40.4y$
Total	$6.8 + 5.8x - 2.87y$	$92.4 + 60.5x - 19.4y$

Government tests by Fernald give the following:

		Bituminous coal	Subbituminous	Lignite	Peat
B.t.u. per pound of coal		12,370	9,910	7,110	8,130
Cubic foot of gas per pound of coal		61.10	39.30	27.70	30.30
Rate of gasification		7.64	11.02	13.28	16.20
B.t.u. per cubic foot gas, standard condensers		151.0	159.0	161.0	175.00
Producer efficiency, cold gas basis		0.75	0.63	0.63	0.65
Fuel analysis, per cent:	Moisture	6.6	15.0	35.7	21.0
	Volatile	32.8	34.3	29.2	51.7
	Fixed carbon	50.6	39.4	27.2	22.1
	Ash	10.0	11.3	7.9	5.2
	Sulphur	2.3	0.9	1.1	0.5
Gas analysis by volume, per cent:	CO_2	9.71	11.16	9.90	12.40
	O_2	0.02	0.12	0.13	
	C_2H_4	0.19	0.20	0.10	0.40
	CO	19.03	17.52	20.86	21.00
	H_2	13.48	14.41	14.30	18.50
	CH_4	2.78	3.64	2.88	2.20
	N_2	54.79	52.95	51.83	45.50

The table on following page gives typical results with various coals in different types of producer.

TYPICAL ANALYSES OF PRODUCER GASES FROM VARIOUS FUELS
(Values in percentage by volume)

Constituents	Up-draft pressure-producer gas				Down-draft producer gas		
	Anthracite	Bituminous coal	Lignite	Peat	Bituminous coal	Lignite	Peat
Carbon dioxide.....CO_2	5.2	9.84	10.55	12.40	6.22	11.87	10.94
Oxygen.....O_2	0.4	0.04	0.16	0.00	0.13	0.01	0.41
Ethylene.....C_2H_4	0.0	0.18	0.17	0.40	0.01	0.00	0.06
Carbon monoxide...CO	22.9	18.28	18.72	21.00	21.05	16.01	16.91
Hydrogen .....H_2	15.3	12.90	13.74	18.50	12.01	14.76	10.19
Methane.....CH_4	1.0	3.12	3.44	2.20	0.49	0.98	0.66
Nitrogen.....N_2	55.2	55.64	53.22	45.50	60.09	56.37	60.83
	100.00	100.00	100.00	100.00	100.00	100.00	100.00

(From MARKS' *Mechanical Engineers' Handbook*)

By-product Plants.—In ordinary producer operation, ammonia formed from the nitrogen in the coal is wasted at the scrubber outlet. In the Mond process, it is recovered as ammonium sulphate. Tar is also reclaimed and the gas goes to the engine very clean and of uniform quality. A deep, low-temperature fuel bed is used.

Construction and Operation.—Shells are of calked steel sheets, made up into vertical cylinders, lined and with water-cooled tops. The base may be tight,

Fuel	Type of engine	Compression, lb. per sq. in.		Heat value average B.t.u. per pound (low value)
		Usual Range	Gage Average	
Natural gas.....	Medium and large size	75–160	120	21,300
Coke-oven gas.....	Large engines (Europe)	105–135	120	14,400
Coal gas.....	Small sizes	75–120	100	17,000
Carbureted water gas.....	Small sizes	75–105	90	7,200
Producer gas.....	All sizes	100–160	130	1,950
Blast-furnace gas.....	Largest sizes	120–190	155	1,500

or water-sealed. Peep-holes and cleaning doors are provided. Mechanical stirring and mechanically moved grates are available. Cleaning requires removal of dust down to about 0.01 gr. per cubic foot of gas and elimination of moisture. Dust removal is effected by dry cleaners which change the direction and reduce the velocity of gas flow, by filters, by tower washers (scrubbers) and by centrifugal washers. In soft-coal plants, special tar-extracting devices must be added. The first cost of entire gas making plant will be about three times

the cost of producers alone. In normal times, blast-furnace gas installations (requiring only engines, buildings and gas cleaning equipment) can be set up for $100 per kilowatt.

GAS ENGINES

Fuels.—Engines using *liquid* fuels are considered below (page 66). Some analyses of typical *gas* fuels are given in the table on p. 62.

The figures in the last column of the table are no index to the value of the gas for power purposes. They do not determine engine efficiency or engine size.

Types of Engine.—Gas engines may be vertical or horizontal, the former being mainly in small sizes. They may be single or double acting: the former generate power only about half as fast, but the pistons and rods require no water cooling. Trunk pistons make for simplicity, but can be used on small engines only. Important engines are multi-cylinder, both tandem and twin arrangements being used. The latter gives smoother running and decreases overall length: it costs more.

Engines are two-cycle or four-cycle according as the sequence of operations is completed in two or four single strokes. Two-cycle engines may be of much simpler construction: they are sometimes practically valveless. They give about 70 per cent more power for their size than four-cycle types. They consume considerably more fuel per horsepower-hour. With a given number and structural arrangement of cylinders, they run more smoothly.

Capacities.—If n = number of cylinders, S = piston speed, feet per minute, a = piston area, square inches, p_m = mean effective pressure, pounds per square inch, the indicated horsepower of a four-cycle single-acting engine is

$$H_I = \frac{p_m a n S}{132000}.$$

The two-cycle engine gives ideally twice this power. For double-acting engines, multiply these figures by 2. If E_M = mechanical efficiency, the brake horse-power is $H_B = E_M H_I$. Values of E_M are given by Lucke as follows:

MECHANICAL EFFICIENCIES

(Lucke, "Thermodynamics")

TYPE OF ENGINE	MECH. EFFICIENCY 4-CYCLE	2-CYCLE
Small or medium single-cylinder, stationary, single-acting	0.87	0.70
Small or medium 2-cylinder, stationary, single-acting	0.84	.....
Small or medium 3-cylinder, stationary, single-acting	0.82	.....
Small or medium 4-cylinder, stationary, single-acting	0.80	.....
Large single-cylinder, stationary, single-acting	0.90	0.70
Large 2-cylinder, stationary, single-acting	0.86	to
Large 4-cylinder, stationary, single-acting	0.84	0.80
Large single-cylinder, stationary, double-acting	0.83	0.75
Tandem 2-cylinder, stationary, double-acting	0.81	0.73
Tandem twin 4-cylinder, stationary, double-acting	0.77	0.69

The following values are given for expected mean effective pressures (p_m) under good conditions:

Fuel	Compression, pounds per square inch, absolute:						
	70	80	90	100	120	140	160
Natural gas, water gas, coal gas	62–82	67–87	70–97	70–100	83–103		
Producer gas	.....	.....	.....	55–80	62–87	65–90	75–90
Blast-furnace gas	.....	.....	.....	.....	60–85	60–90	70–95

Efficiency.—The maximum ideal efficiency (Otto cycle) is

$$E_I = 1 - \left(\frac{14.7}{P_c}\right)^{0.286}$$

where P_c = compression pressure, pounds per square inch, absolute. The probable actual efficiency lies between $0.45E_I$ to $0.55E_I$ for most fuels, and from $0.40E_I$ to $0.50E_I$ for blast-furnace gas. The heat consumption per indicated horsepower per minute is therefore $42.42/0.5E_I = 84.84/E_I$ B.t.u., approximately. The weight of gas consumed per indicated horsepower per minute is $84.84/BE_I$ lb., where B = low-heat value of gas, B.t.u. per pound.

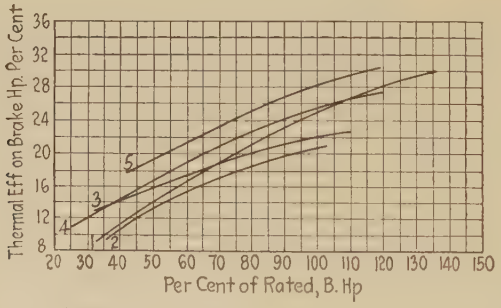


FIG. 28.—Efficiency curves of internal-combustion engines.[1]

Efficiency varies chiefly with the compression and compression is limited by the danger of preignition. Gases high in hydrogen content ignite readily and must be used at low compression. Hence, the gases of high heat value are in general those least useful for engines.

The variation of efficiency with load depends on the method of governing. The hit-and-miss method of governing is probably most economical of fuel but gives irregular action and is used only on small engines. Variation of quantity of combustible mixture varies the compression with the load and hence decreases efficiency seriously at light loads. Variation of mixture strength leads to less change of compression but introduces other losses at light loads, due to impaired combustion. Misfires may be produced. The diagram shows in all cases maximum efficiency at maximum load: a characteristic of the gas engine, which has no overload capacity beyond its output of best economy. All of these engines were governed by adjustment

[1] Curve 1, Deutz single cylinder, illuminating gas; Curve 2, Westinghouse, 3-cylinder vertical, natural gas; Curve 3, Dentz, producer gas; Curve 4, Güldner, single cylinder, producer gas; Curve 5, Nürnberg, blast-furnace gas. (Cf. Marks, *Mechanical Engineers' Handbook*, 1st. ed., p. 1029.)

of quantity or strength. The following table illustrates the same characteristic as the chart.

RELATIVE INCREASE OF FUEL CONSUMPTION PER B.HP. PER HOUR AT PARTIAL LOADS

	Full-load	¾ load	½ load	¼ load
Natural-gas engine	1.00	1.05–1.20	1.20–1.50	1.75–2.00
Illuminating-gas engine	1.00	1.05–1.20	1.20–1.50	1.75–2.00
Producer-gas plant (engine and producer)	1.00	1.15–1.25	1.45–1.60	2.30–2.70
Gas producer	1.00	1.04–1.06	1.10–1.15	1.30–1.40
Blast-furnace gas engine	1.00	1.05–1.20	1.20–1.50	1.75–2.00

(From MARKS' *Mechanical Engineers' Handbook*)

Losses.—The chief heat losses in gas engines are those to the jacket water and the exhaust: each of these losses is roughly equal to the heat converted into power. The jacket water outflow temperature is rarely above 160°, so that this

HEAT BALANCES OF GAS AND OIL ENGINES

(Per cent of heat of combustion.—Lucke, "Thermodynamics")

Engine	I.hp.	B.hp.	Friction[a]	Jacket	Exhaust	Radiation and unaccounted for
General (Mathot)	33.0	28.0	5.0	36.0	31.0[b]	.....
300-hp. engine at 197 hp	43.5	33.5	10.0	34.3	24.1	1.9[c]
300-hp. engine at 294 hp	45.8	32.2	13.6	31.8	23.9	1.5[c]
300-hp. engine at 335 hp	41.5	30.9	10.6	33.8	24.8	0.1[c]

[a]Including pump work. [b]Including radiation, etc. [c]Excess.

heat is largely unavailable for further use. If the water is clean, some of it may be used for boiler feed or for heating processes: or its temperature may be increased to fit it for warming buildings or for process work. The exhaust gas temperature is 600 to 700°, but the gases part with their heat at a slow rate, since dry gas is a poor conductor. Devices for transferring this heat to water, steam, etc., must therefore have ample surface.

Details.—*Ignition* is now universally electrical. Large engines employ low-tension direct-current mechanical make and break. High-tension jump-spark ignition is used for smaller sizes. The ignition should usually start at the center of the combustion space, but in large cylinders a plurality of igniter electrodes will be required. Starting is by hand, by motor geared to the flywheel, or by compressed air. Air-cooling is used only for very small engines. The cooling water required is about 10 gal. per brake horsepower per hour. From 10 to 20 lb. water pressure is sufficient unless pistons are cooled (double-acting

engines). A gas receiver is desirable on the supply side of the throttle. Mufflers are frequently required. They should be much larger than they are commonly built: say 15 times the volume of piston displacement per stroke.

Cost of Gas Power.—The gas engine is essentially more efficient than the steam engine. Generally, a complete gas plant (producers and engines) costs considerably more than a steam plant of the same capacity. Hence the best application of gas power is where fuel costs are high (for both gas and steam) and where the load is steadily maintained. Fixed costs are paid every hour: fuel costs merely during the hours the engine is operated. Incidentally, the question of reliability may sometimes favor steam: steam has also the advantage of lending itself better to heating and process demands.

Power Costs in a 450-kw. Producer-gas Power Plant

(Running on bituminous coal)

	Minimum load factor	Normal load factor	Maximum load factor
Load factor, per cent	50	80	100
Output, kw.-hr. per day	5,000	8,000	10,000
Coal consumption, pounds per kilowatt-hour	2.175	1.940	1.880
Fuel cost cents per kilowatt-hour	0.250	0.223	0.216
Labor cost, cents per kilowatt hour	0.280	0.175	0.140
Supplies and repairs, cents per kilowatt-hour	0.147	0.092	0.074
Total operating cost, cents per kilowatt-hour	0.677	0.490	0.430
Fixed charges, 15 per cent., cents per kilowatt-hr.	0.718	0.449	0.359
Total power cost, cents per kilowatt-hour	1.395	0.939	0.789

(From Marks' *Mechanical Engineers' Handbook*)

The plant of the foregoing table consists of three 150-kw. three-cylinder, vertical Westinghouse, single-acting, four-cycle, engines, with producers. The coal averaged 13,500 B.t.u. per pound and cost $2.30 per short ton delivered. The load is carried 24 hr. daily, 6½ days weekly, at a labor cost for two shifts of $14 per day. The cost of the entire plant (producers, engines, generators and auxiliaries) was $180 per kilowatt of capacity.

OIL ENGINES

Classification.—Internal combustion engines using liquid fuel may be two-cycle or four-cycle. The two-cycle engine compares more favorably with the four-cycle with regard to efficiency than when gas is the fuel, but is nevertheless inferior. Liquid fuel engines may be supplied with a combustible mixture of fuel and air, or the air and fuel may be separately fed. Engines using carburetors belong to the former class. Those of the latter class may receive fuel gradually during the suction and (or) compression strokes, as in many "low pressure" (hot cap or vaporizer) types: or the oil may be injected late in the compression stroke. If compressions exceed 200 lb., the oil must then be forced

in by compressed air at high pressure. If it is forced in suddenly, the engine is of the "high-pressure" or "semi-Diesel" type, although it uses the Otto rather than the Diesel cycle. If the oil delivery is gradual, the Diesel engine results: in either case maximum pressures of 500 to 600 lb. are realized.

Carburetor Engines.—The gasoline engine is the type of all of these, the fuel being mixed with air by vaporization and agitation, and the mixture drawn into the engine by piston suction. The heat value of gasoline is close to $18{,}320 + 40(B - 10)$ B.t.u. per pound where B = Baumé hydrometer reading. The diagram shows the low vaporizing temperature of gasoline at a given pressure. Kerosene, as indicated, vaporizes with much less readiness. It therefore requires a heated carburetor. This implies complication in starting. It also requires a limitation of compression or water injection to prevent preignition. The alcohols vaporize with moderate ease. Ethyl alcohol of 95 per cent volumetric strength has a heat value of 10,880 B.t.u. per pound (low). For kerosene, B.t.u. per pound = $18{,}440 + 40(B - 10)$.

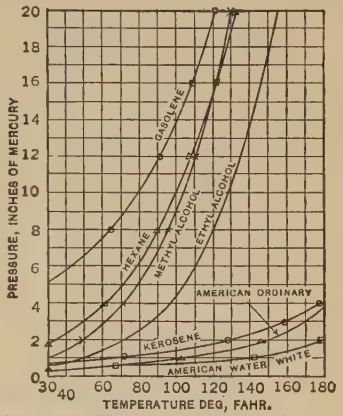


FIG. 29.—Vapor pressures of liquid fuels.

USUAL COMPRESSION PRESSURES FOR INTERNAL-COMBUSTION ENGINES

Fuel	Type of engine	Range of compression, lb. per sq. in. gage	Average compression in lb. per sq. in. gage
Gasoline	Automobile	45–95	65
Gasoline	Stationary	60–105	70
Kerosene	Vaporized before entering cylinder	45[1]–85[2]	65
Alcohol	Vaporized before entering cylinder	120–210	150

[1] With hot mixture without water injection.
[2] With water injection.

(From MARKS' *Mechanical Engineers' Handbook*)

All of these engines are of importance in small sizes only, where the high cost of fuel used can be tolerated. They have jump-spark or make-and-break electric ignition. Mean effective pressures for gasoline are 70 to 85 lb. at 65 lb. compression (absolute) and 85 to 95 lb. at 100 lb. compression. For kerosene, 50 to 65 lb. mean effective pressure is realized at the lower compression, 70 to 85 lb. at 115 lb. compression. The equation for horsepower given on page 63 applies to these engines. The ideal efficiency is also as there indicated, and the actual efficiency usually about half the ideal. Jacket and exhaust heat constitute the two large losses. Air-cooling is practicable in small sizes and unimportant applications.

Hot Cap Engines.—These have an unjacketed cap or vaporizer into which the liquid fuel is injected. The cap is externally heated for starting. Where the fuel injection is not "timed," ignition depends on compression as well as cap temperature and its timing is apt to be uncertain, especially at variable loads. Compressions are generally from 50 lb. upward. High compressions lead to high

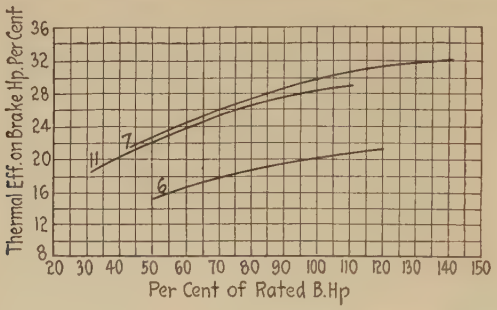


Fig. 30.—Efficiency curves of internal combustion engines.[1]

mean effective pressures and are associated with high economies. A compression of 50 lb. will give a fuel rate of about 1.1 lb. per brake horsepower-hour: 300 lb. compression will give about half this fuel rate. Kerosene hot-cap engines ae 250 to 500 r.p.m. use 30 to 75 lb. compression, averaging 60 lb. (gage). The Hornsby-Akroyd engine with fuel oil of high grade (about 35°) uses 45 lb. Tht Franchetti (De La Vergne FH) engine uses nearly 300 lb., with oils as low as 20°. For an engine of Hornsby-Akroyd type, fractional load fuel rates are about as follows:

	Full Load	Three-fourths Load	One-half Load	One-fourth Load
Relative fuel rates.....	1.0	1.15–1.35	1.45–1.70	1.90–2.30

The low-pressure engines of this class are used mainly with kerosene or solar oil of about 35°. Heavier oils must be more finely atomized than is possible by pump injection against low pressures. With high pressures, an air blast is used to deliver the oil. This implies an air pressure about twice the maximum existing in the cylinder, often 1,000 lb. or more. The air compressor must be multi-stage with ample cooling, and air storage tanks are necessary for the supply of fuel when starting. Low-grade oils will require heating in order that they may flow readily.

With fuel injection by pump, the engine may be governed either by varying the pump stroke or by by passing a portion of its discharge back to the supply tank.

Diesel Engines.—In the Diesel engine, air only is compressed to about 500 lb. pressure, attaining a temperature which is sufficient to ignite the incoming fuel. The fuel is sprayed in by compressed air at a controlled rate. The maximum pressure rarely exceeds 550 lb. At normal load, oil delivery continues for about one-tenth the stroke, giving a mean effective pressure around 100 lb., and a fuel rate of about 0.5 lb. per brake horsepower-hour. Theoretically, the fuel rate should improve at light loads. Actually (chiefly on account of friction losses) it increases like that of any other engine, though not as rapidly: three results being as follows:

[1] Curve 6, Swiderski, single-cylinder alcohol; Curve 7, Dentz, single-cylinder alcohol; Curve 11, Bánki, gasoline. (Cf. Marks, *Mechanical Engineers' Handbook*, 1st. ed., p. 1029.)

		Full-load	Three-fourths load	One-half load	One-fourth load
Relative fuel rates	*A*	1.0	1.02–1.15	1.07–1.25	1.40–1.90
	B	1.0	1.05	1.16	.....
	C	1.0	1.06	1.17	.....

On account of the high pressures used, Diesel (and semi-Diesel) engines cannot be built with very large cylinders. They are heavy and costly. For large powers, a large number of cylinders is necessary. This again implies great bulk, weight and cost. The air compression plant is large and complicated.

Place of the Oil Power Plant.—While oil engines are expensive, and first cost varies in the same way as efficiency, the engine constitutes almost the whole plant, because (almost alone among heat engines) it uses raw fuel.

Its high economy goes far to offset the usually high cost of fuel per British thermal unit. Maintenance costs are high, and miscellaneous operating costs not always low Reliability has not been thoroughly demonstrated. Diesel and semi-Diesel engines are complicated and employ high pressures. Low-pressure oil engines are somewhat uncertain as to ignition. The assured field at present seems to be for small isolated powers, although large installations and some marine plants have been successfully operated.

The expression given for horsepower on page 63 applies both to hot-cap and Diesel engines. Losses due to jackets and exhaust are about as for gas engines. These engines are always water-cooled. Engines of low pressure type cost two to three times as much as steam turbines: high-pressure and Diesel engines are about twice as costly per horsepower as low-pressure. There has been little attempt to use asphaltum base (Western) oils in internal-combustion engines.

WATERPOWER

Water Wheels.—The older types of wheel give maximum efficiencies up to 70 to 90 per cent when used under the particular conditions for which they are suitable: *i.e.*, for heads up to 70 ft. and powers not exceeding 75 hp. Important wheels are turbines. These are of the tangential (impulse) or reaction type. Tangential turbines use the velocity of water to produce power, the head being wholly converted to velocity in open-end nozzles. They run at low peripheral speeds for given heads and are therefore the type best adapted to high heads (300 to 3,000 ft.) and small quantities of water. Their construction is simple. The runners carry double-lobed buckets toward which the jets are directed and the stream is deflected about 90° on both sides of the "splitter."

Reaction turbines utilize the pressure, velocity and weight of water. The Francis type universally used in this country receives the water through stationary guide vanes which carry it radially inward and partially convert the head into velocity. The discharge is parallel with and close to the shaft. Such turbines are adapted for heads of 5 to 600 ft. They may be vertical or horizontal: the former are better adapted for direct-connection where water levels fluctuate, but require thrust bearings. For heads of 5 to 50 ft., the turbines are often set in an open flume. For higher heads, they are encased, the spiral form of casing being best. Some vertical low-head turbines

have the casing formed in concrete. Plate-steel casings with either end or top inlets are commonly used. For the higher heads, cast iron casings are most popular. Either type of wheel may employ two runners on one shaft.

Characteristic.—If h = head available, feet, n = revolutions per minute desired, P = horsepower of each turbine at the runner, the *specific speed* or *characteristic* is $n_s = n\sqrt{P} \div h^{5/4}$. This is that ideal speed (revolutions per minute) which a similar wheel would have if operating under 1 ft. head and then developing 1 hp. Each design has its own specific speed: when size is changed by change of scale (all dimensions changed proportionately) the specific speed does not change.

The best values of n_s are from 1 to 4 for tangential and from 30 to 60 for reaction turbines. Allowable values for the former are up to 8 (per nozzle) and for the latter from 10 to 100, but extreme values lead to low efficiencies. If the available head and required power and speed lead to abnormal values of n_s, some adjustment is possible by varying the number of units. Thus if 2,550 hp. were required at 750 r.p.m. with 100 ft. head, n_s would be 120, a value not permissible. By using two wheels (or runners) of 1,275 hp. each, an allowable value results. In some cases the speed could be changed. The equation for specific speed leads to the following forms:

$$\frac{n_1 P_1^{1/2}}{h_1^{5/4}} = \frac{n_2 P_2^{1/2}}{h_2^{5/4}}$$

At constant speed, $\frac{P_1}{P_2} = \left(\frac{h_1}{h_2}\right)^{5/2}$: power varies much more rapidly than head.

At constant head, $\frac{P_1}{P_2} = \left(\frac{n_2}{n_1}\right)^2$: power varies inversely as the square of the speed.

At constant power, $\frac{n_1}{n_2} = \left(\frac{h_1}{h_2}\right)^{5/4}$: speed varies much more rapidly than head.

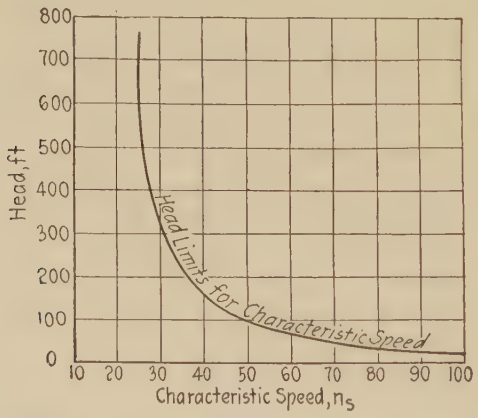


FIG. 31.—Practical limits for turbine runners.

These relations hold for wheels of a given series. They do not include the effect of varying efficiency.

The accompanying chart shows the low limits practicable for specific or characteristic speed at various heads. If speeds are made lower, the runner passages become very long and narrow, the diameter great and the cost high.

A given value of n_s implies a given value of m, the peripheral coefficient, or ratio $u \div \sqrt{2gh}$ where u = peripheral speed of runner rim, feet per second, at the actual speed. Thus, for Allis-Chalmers reaction wheels:

Type	A	B	C	D	E	F	G	H
n_s.....	13.55	20.30	29.40	40.70	51.7–60.5	72.00	82.00	92.50
m.....	0.585	0.625	0.665	0.70	0.75	0.80	0.77	0.815

If a wheel of type C were used under 400 ft. head at 1,200 r.p.m., the power developed at the runner would be $(29.4 \times 400^{5/4} \div 1{,}200)^2 = 1{,}920$. With an efficiency of 0.80, the brake horsepower is $1{,}920 \times 0.8 = 1{,}536$. The volume of water required per second is $Q = (550 \times P)/62.4h$ cu. ft., or 42.4 cu. ft. for the assumed conditions. The peripheral speed is $0.665\sqrt{2gh} = 0.665 \times 20 \times 8.02 = 106.7$ feet per second and the wheel diameter is $106.7 \times 12 \div 20\pi = 20.4$ in. Runners of high specific speed are usually less efficient at normal load than those of low specific speed, but hold up better in power and efficiency at over-gate flows, and are therefore best adapted for variable heads, especially in low-head plants. The efficiency at part-load is of no importance unless water is stored, and storage in low-head plants is rare. High specific speeds imply large wheel diameters for a given head and rotative speed: the range of diameters (inches) being from $1{,}070\sqrt{h}/n$ to $1{,}530\sqrt{h}/n$ for specific speeds from 13½ to 92½. Since $P = n_s^2\, h^{5/2} \div n^2$, high specific speeds also increase the power of a wheel to be operated at fixed head and rotative speed.

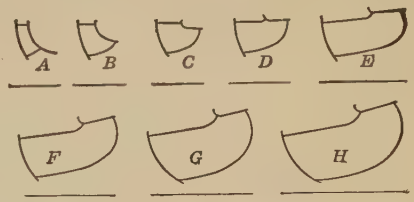


Fig. 32.—Typical runner-blade profiles.

Tangential (Impulse) Wheels.—The tangential water wheel is somewhat less efficient than the reaction turbine at full load, but its efficiency at underloads is relatively (and often absolutely) greater. The wheel diameter is usually about $840\sqrt{h} \div n$ in., or is smaller for a given head and speed than that of a reaction wheel. The jet diameter is usually about 1/12 the wheel diameter (1/20 to 1/10) and rarely exceeds 8 in. An approximate

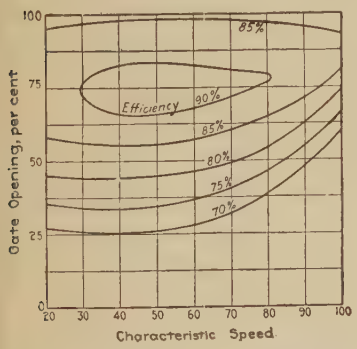


Fig 33.—Relations between characteristic speed, efficiency and gateage of reaction turbines.

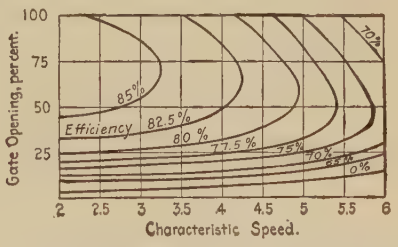


Fig. 34.—Relations between characteristic speed, efficiency and gateage of impulse wheels.

expression for power (at the runner) is $P = d^2h^{3/2} \div 2.12$ where d = jet diameter, inches. A more exact expression is $P = n_s^2h^2d/840n$.

Governing.—Reaction wheels are governed by varying the gate opening. The plain cylinder gate is now generally replaced by wicket-type gates or guide vanes. Outside gate gears permit of lubrication and are used with high head wheels or with single-runner concrete spiral cases. Inside gears are submerged in the wheel pit and working surfaces must be bronze-lined. The vanes are of cast iron or steel.

Impulse wheels are governed by needle nozzles (changing the size of the jet) or deflection nozzles (diverting some of the water from the wheel). The former method is the much more economical of water, but as stated, economy is not always important. It cannot be used without auxiliary devices, where the penstock is long and the inertia of the water great. In such cases, a sudden check of flow may actually increase rather than decrease the wheel output. Combination nozzles are often used: the first action being a deflection, followed by a gradual swinging back of the nozzle to normal positon and a gradual. closing of the jet needle.

The power required to operate gates or nozzles is so great that indirect governing is general. Hydraulic pressure is generally employed, the fluid being distributed through a valve actuated by a fly-ball governor to a cylinder which operates the final control. Large electric generators can be given adequate regulation without particular difficulty: for small direct-connected plants the governing must be worked out with some care.

Construction.—Runners of reaction turbines are of cast iron, bronze, cast steel and composite of plate steel in cast hubs. The material used is determined chiefly by manufacturing considerations. Thrust bearings for horizontal units are of the plain collar type. For vertical turbines, ball, roller or oil-pressure (step) bearings are commonly used. Impulse turbine buckets are of bronze, cast iron or cast steel: cast iron being used only for the lower heads. Nozzles are cast iron or cast steel, usually with removable bronze tips and steel needles. Interior surfaces should be polished smooth. The housing is usually cast iron, sometimes with steel-plate covers.

Draft Tube.—By using a closed tube for the discharge of a reaction turbine, the machine may be elevated above the tail water level and the velocity of discharge water largely utilized, the water doing work by suction. The end of the tube must be submerged, and it should flare toward the end. Bends introduce considerable losses. They cannot be avoided with double runner turbines where one draft tube serves both wheels. The turbine should never be more than 20 ft. above the tail water level.

Costs.—Even excepting cases where water rights are purchased, the cost of the water wheel is only a small part of the total cost of a waterpower plant. Wheels plus electric generators cost in normal times from $6 to $30 per horsepower, for sizes from 10,000 hp. down to 500 hp. The form (vertical or horizontal) has little effect on cost. High heads reduce costs, especially for the smaller sizes: such heads being usually applied to impulse (tangential) wheels.

Determination of Water Available.—The discharge of a stream is ascertained by measuring the water velocity at various points of the width and at such depth as to indicate the approximate mean velocity, and by considering mean velocities

of various small transverse sections of definite area, running from surface to bed of the stream. Discharge is expressed in cubic feet per second. It usually varies more rapidly than the transverse area of the stream: high water increasing the discharge more than it does the area. A plot of discharge against relative surface level permits of quick estimates of variations in discharge.

While large discharges are accompanied by high surface levels above the dam, the transverse area of the stream below the dam is generally so contracted that large discharges increase the tail water level more than they do the head water level. The available head is the difference of head between head water and tail water. Hence head decreases as flow increases, and the adaptability and capacity of the wheels for varying heads and flows must be considered.

The hydrograph, or plot of discharge against time, and the power curve, or plot of *discharge* × *head* against time, show the fluctuations in flow and power over a period of months or years, and determine what flow or power may be relied on as fixing the desirable capacity of installation. The *mass-flow* curve plots flow against time cumulatively. A tangent drawn to the lowest point of this curve from the origin shows the average flow which may be depended on *with unlimited storage* of water. The possibility of storage depends largely on whether there are rights below the site of power development in question which make interference with the natural stream flow unlawful. In any case, storage can scarcely be considered except where the topography of the country favors it: *i.e.*, in high-head plants. *Pondage* (moderate storage, of a few hours' discharge only) may be used even in low-head plants where the load for short periods during a day exceeds that corresponding with the flow: but even pondage is impossible if the stream flow must be left unimpaired.

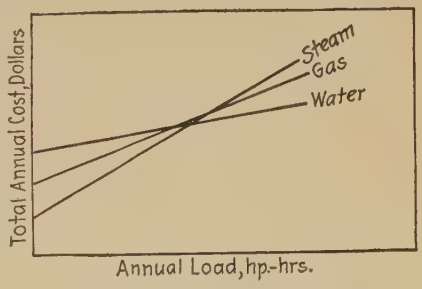


FIG. 35.—Load-plant cost curves.

A really valuable hydrograph or mass-flow curve requires observations extending over several years, and these are not always available. A rough estimate of flow variations may then be obtained from rainfall records. If the monthly rainfall over the known drainage area of the stream has been recorded, the maximum possible monthly stream flow is the product of the rainfall in feet by the drainage area in square feet. This value will be most closely approximated in a wooded rocky country where practically all rainfall is carried off by streams. In other instances as in sandy deserts the rainfall may practically all percolate into the soil or be lost by evaporation, so that the probable ratio of run-off to rainfall must be carefully estimated for the plant in question, by considering the nature of the soil and vegetation, humidity, etc.

Cost of Development and Generation.—A waterpower project may include dams, power-house buildings, head race or penstock, tail race, gates and sometimes pondage or storage. Approximately $100 per horsepower for low-head plants without storage and twice this or more for high head plants including storage may be counted on. Waterpower plants are in general the most expensive of all types to install and the least expensive to operate: the cost of power per horse-power-year ranging from $10 to $35. As with all power plants, total cost per year tends to vary linearly with the annual-output: *i.e.*, a good load factor lowers the cost per horsepower. The curves of the accompanying diagram are typical. The intercepts on the vertical axis are proportional to the relative

costs of installation. The slopes are proportional to relative costs of operation.

POWER TRANSMISSION IN FACTORIES

Belting.—For very dry places, oak-tanned is the preferred leather belting. For damp places use rawhide or chrome-tanned or rubber belt. The last should not be used where it may be subjected to oil or grease. Canvas has about the same strength as leather belting. Riveted or laced joints greatly reduce strength: cemented joints do not.

HORSE-POWER TRANSMITTED BY TURNED STEEL LINE SHAFTING

Diameter of shaft, inches	Number of revolutions per minute										
	100	150	200	250	300	350	400	450	500	550	600
1½	3.7	5.6	7.5	9.4	11.2	13.1	15	16.9	18.8	20.5	22
1⅝	4.8	7.1	9.5	11.9	14.3	16.6	19	21.0	24.0	26.0	28
1¾	5.9	8.9	11.9	14.9	17.9	21.0	24	27.0	30.0	33.0	36
1⅞	7.3	11.0	14.7	18.3	22.0	26.0	29	33.0	37.0	41.0	44
2	8.9	13.3	17.8	22.0	27.0	31.0	35	40.0	44.0	48.0	53
2⅛	10.6	16.0	21.0	27.0	32.0	37.0	43	48.0	53.0	58.0	64
2¼	12.6	19.0	25.0	32.0	38.0	44.0	51	57.0	63.0	69.0	76
2⅜	14.9	22.0	30.0	37.0	45.0	52.0	60	67.0	74.0	81.0	89
2½	17.4	26.0	35.0	43.0	52.0	61.0	69	78.0	87.0	96.0	104
2⅝	20.0	30.0	40.0	50.0	60.0	71.0	80	90.0	100.0	110.0	120
2¾	23.0	35.0	46.0	58.0	69.0	81.0	92	104.0	115.0	125.0	138
2⅞	26.0	40.0	53.0	66.0	79.0	92.0	105	119.0	132.0	145.0	158
3	30.0	45.0	60.0	75.0	90.0	105.0	120	135.0	150.0	165.0	180
3⅛	34.0	51.0	68.0	85.0	102.0	119.0	136	152.0	170.0	187.0	203
3¼	38.0	57.0	76.0	95.0	114.0	134.0	153	172.0	191.0	210.0	229
3⅜	43.0	64.0	85.0	107.0	128.0	150.0	171	192.0	213.0	234.0	256
3½	48.0	72.0	95.0	119.0	143.0	167.0	190	214.0	238.0	262.0	286
3⅝	53.0	79.0	106.0	132.0	159.0	185.0	211	238.0	265.0	291.0	317
3¾	59.0	88.0	117.0	146.0	176.0	205.0	234	264.0	293.0	322.0	351
3⅞	65.0	97.0	129.0	161.0	194.0	226.0	258	291.0	322.0	354.0	387

NOTE.—For cold rolled shafting up to 5 in. add 30 per cent. For turned steel head shafts, subtract 30 per cent. For cold-rolled steel head shafts, subtract 10 per cent. For transmission shafts of turned steel without pulleys add 80 per cent. For transmission shafts (without pulleys) of cold-rolled steel, add 125 per cent.

If T_1 and T_2 are tensions in pounds on the pulling and slack sides of a belt, respectively,

$$\log \frac{T_1}{T_2} = 0.0076\, fa \text{ for low speeds,}$$

$$= 0.0076\, f(1 - x)a \text{ for high speeds,}$$

$$x = wv^2 \div 2.683t, \text{ and}$$

$$\text{horsepower} = (T_1 - T_2)v \div 550.$$

In the above

f = coefficient of friction = { 0.30 for leather belts on cast iron pulleys. 0.45 for leather belts on wood pulleys. 0.55 for leather belts on paper pulleys.

a = arc of contact, degrees,

w = weight of belt material per cubic inch = 0.035 lb. for leather,

v = speed of belt, feet per second, preferably from 67 to 75, and

t = working tension, pounds per square inch (300 for leather).

The value of f is increased by dressings, dampness or pulleys with cork inserts. Recommended values of $(T_1 - T_2)$ are 35 for single leather, 50 for double leather, 22 lb. per ply for canvas: all *per inch of width.* Pulleys used with double leather belts should not be less than 12 in. diameter.

Shafting.—For large installations, it pays to calculate shafting sizes closely, considering both transmission and bending stresses. The accompanying table

HORSEPOWER TRANSMITTED BY TURNED STEEL LINE SHAFTING

Diam. of shaft, in.	Number of revolutions per minute										
	100	125	150	175	200	225	250	275	300	400	500
4	71.0	89.0	107.0	125.0	142.0	160.0	178	196.0	213.0	284.0	356
4¼	85.0	107.0	128.0	149.0	170.0	192.0	213	234.0	256.0	341.0	426
4½	102.0	127.0	152.0	178.0	203.0	228.0	253	278.0	305.0	405.0	507
4¾	119.0	149.0	179.0	209.0	238.0	268.0	298	328.0	357.0	476.0	595
5	139.0	174.0	208.0	244.0	278.0	313.0	347	382.0	417.0	557.0	695
5¼	161.0	201.0	242.0	281.0	322.0	362.0	403	443.0	483.0	644.0	805
5½	184.0	230.0	277.0	322.0	369.0	415.0	461	507.0	553.0	738.0	922
5¾	211.0	264.0	317.0	369.0	422.0	475.0	528	580.0	633.0	844.0	1,055
6	240.0	300.0	360.0	419.0	480.0	540.0	600	660.0	720.0	960.0	1,200
6¼	271.0	339.0	407.0	473.0	542.0	610.0	678	745.0	813.0	1,084.0	1,355
6½	305.0	382.0	459.0	535.0	611.0	687.0	764	840.0	917.0	1,222.0	1,528
6¾	341.0	427.0	513.0	598.0	682.0	767.0	853	938.0	1,023.0	1,364.0	1,705
7	381.0	476.0	573.0	667.0	762.0	857.0	953	1,048.0	1,143.0	1,524.0	1,905
7¼	423.0	529.0	636.0	742.0	847.0	953.0	1,059	1,264.0	1,270.0	1,693.0	2,116
7½	468.0	586.0	704.0	822.0	938.0	1,055.0	1,173	1,290.0	1,406.0	1,875.0	2,344
7¾	516.0	646.0	776.0	904.0	1,033.0	1,163.0	1,293	1,422.0	1,550.0	2,066.0	2,583
8	568.0	712.0	855.0	998.0	1,138.0	1,280.0	1,423	1,565.0	1,707.0	2,275.0	2,844
8½	681.0	853.0	1,025.0	1,197.0	1,364.0	1,535.0	1,707	1,878.0	2,047.0	2,728.0	3,411
9	809.0	1,013.0	1,217.0	1,421.0	1,620.0	1,823.0	2,027	2,230.0	2,430.0	3,240.0	...
9½	951.0	1,191.0	1,431.0	1,671.0	1,904.0	2,143.0	2,382	2,620.0	2,858.0	.....	...
10	1,111.0	1,388.0	1,666.0	1,944.0	2,222.0	2,500.0	2,778	3,055.0	3,333.0	.....	...

(From MARKS' *Mechanical Engineers' Handbook*)

is a guide. Mill shafts run from 150 to 400 r.p.m. Cold-rolled shafting has the disadvantage of being apt to spring when keyways are cut. The maximum spacing of bearings in feet should be $10d^{3/2}$ to $6d^{3/2}$ according as the shaft is bare or carries the usual number of pulleys, placed near bearings: d being the shaft diameter in inches. Hangers are obtainable in various "drops" for ceiling,

wall or floor attachment. The following percentage efficiencies are given by Kimball and Barr:

Common bearing (singly)	96–98	Worm gear: varies with thread angle	
Do., long lines of shafting	95	Cast teeth	92
Roller bearings	98	Cut teeth	95
Ball bearings	99	Belting	96–98
Spur gear, including bearings:		Pin-connected chains (bicycle)	95–97
Cast teeth	92	High-grade transmission chains	97–99
Cut teeth	95		

Rope Drives.—Transmission ropes are usually cotton, hemp or manila, three to six strand, according to size. Cotton is most flexible but weakest. Pulley diameters should be at least 36 times the rope diameter. Rope diameters above 1¾ in. should be avoided. The rope may be continuous or a series of ropes may run side by side. The latter system is more assured against total breakdown, is flexible and cheaper to maintain: but it requires many splices and is not adapted for vertical transmission, and provision should be made for equalizing stresses. Efficiencies vary from 0.55 to 0.95 according to the complexity of the drive. Ropes are cheaper than belts but belt pulleys cost less than rope sheaves. Wire ropes are made up of strands, reversely twisted to form the rope. Standard rope has a hemp core. Large sheaves must be used, especially for high speeds.

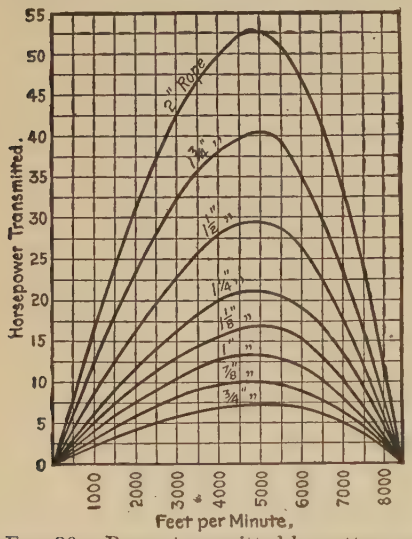


Fig. 36.—Power transmitted by cotton rope.

Hydraulic power transmission is used for slow-moving, powerful machinery requiring accurate control, or where load factors are very low. Efficiencies of hydraulic operation range up to 0.85, the chief loss being that due to the friction at cup leathers used for plunger packings. For direct steam *pumps* having the same number of fluid cylinders as steam cylinders, the ratio of areas of steam and fluid cylinders is $\frac{A_s}{A_w} = \frac{p_w}{p_s e}$, where p_w and p_s are fluid and steam pressures, pounds per square inch, and e = mechanical efficiency of pump, say 0.75. For double-acting cylinders, the volume of fluid discharged per minute is $V = AwSe_v$ where S is piston speed, inches per minute, and e_v is the volumetric efficiency, 0.90 or more for new pumps. The volume is half this for single-acting cylinders. The power required to drive the fluid cylinder is $Vwp_w \div 14{,}300e$ hp. where w = weight of 1 cu. in. of fluid, pounds. Piping should be designed for a fluid velocity of 3 to 5 ft. per second, or a pressure drop of 10 lb. per square inch per mile. A velocity of 100 ft. may be allowed through control valves. Non-freezing fluids must often be used. Double extra-strong wrought

pipe is generally used, with special hydraulic valves and fittings. For screwed-end work, fittings may be forged-steel blocks. For large flanged work, cast-steel fittings with tongued and grooved flanges give good service.

Storage of fluid under pressure may be provided by an accumulator or sheet-iron tank filled with a heavy material like hammer-scale, and attached to the top of a plunger which floats in a cylinder. If d = diameter of plunger, in., h = rise of accumulator, in., the volume of fluid stored is $\frac{\pi}{4}d^2h$ and the weight of the accumulator is $\frac{\pi}{4}p_w d^2$ lb. Pump regulation is effected by trips which are thrown when the accumulator rises or falls. The losses through the accumulator should not exceed 5 per cent.

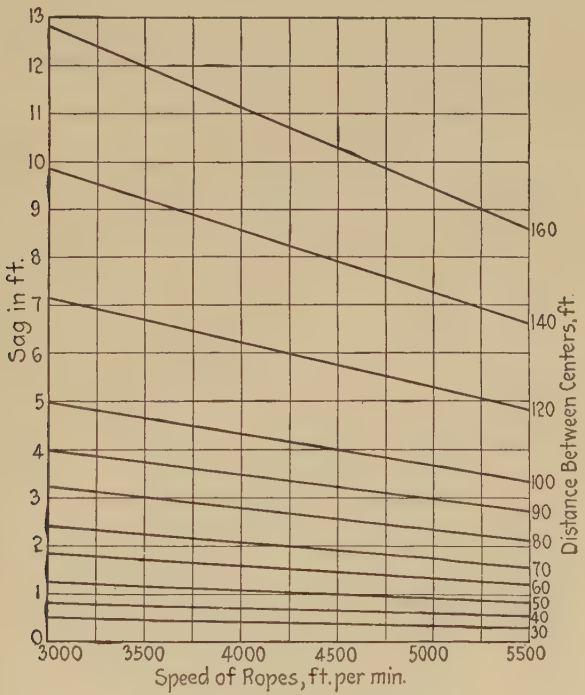


Fig. 37.—Rope-drive constants.

In all cases, the pressure used should be the lowest which will do the work. This saves first cost, power and time. In many instances a part of an operation can be performed at low pressure, the finish only requiring high pressure. This dual system may be provided by a complete duplication of pumps, accumulators and piping, with manual or automatic transfer valves, or intensifiers may be used at points of fluid consumption. These are pumps using the low-pressure fluid as the actuating medium in what corresponds to the "steam" cylinder, to produce a local excess pressure.

Where part of the equipment requires large drafts of fluid (as for instance the flanging presses in a boiler shop) it is permissible to use a dual system with only a slight difference in pressures. The higher pressure fluid goes to supply the steady load and the surplus is vented to the low-pressure system.

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SECTION II

MATERIAL HANDLING—TRANSPORTATION OF SOLIDS

By Reginald Trautschold, M.E.[1]

The Problem.—The transportation of solids by mechanical or semi-mechanical means resolves itself into an economic question in which the three dominant factors are: (1) Material, its characteristics and the quantity to be transported; (2) speed, or time consumed in the operation; and (3) equipment and operation costs. The material to be handled is a known quantity and the time consumed in the operation is subject to close regulation by employing equipment of suitable capacity operated at its most effective speed for handling the class of material, so the handling equipment proper to employ is dependent upon the cost of the equipment and the gross expense entailed in its operation—the net operating cost. No fixed rules can be advanced or definite recommendations made as to the best type of equipment to employ, for rarely are the controlling factors the same in even similar handling problems.

Distinguished from general transportation, which embraces the questions of rail, water and motor-truck transportation over long distances, material handling in the present chapter will be confined to the economic transportation of materials in and about the industrial plant or mine and over comparatively short distances—the conveying and elevating of material by the assistance of mechanical aids.

Classification of Equipment.—Equipment for the mechanical handling of materials may be classified as the carriers which carry the material in a limited but continuous stream (endless conveyors and elevators), and the hoists, trucks and railways which carry more concentrated, and frequently more compact, loads more or less intermittently. This classification will be adhered to so far as practical to simplify a logical selection of handling equipment for the industrial establishment and consideration of equipment will parallel the passage of the material, commencing with its delivery.

Economic Comparisons.—Power being required to operate practically all types of conveying and elevating machinery, the power consumption in horsepower or kilowatts per hour as well as the initial and gross operating costs of the equipment will serve as the basis of economic measurement, the relative net cost of operation being then a simple matter of comparison between suitable systems of transportation.

LOCOMOTIVE CRANES

Characteristics.—Self-propelling, mobile cranes of the locomotive type are ordinarily mounted on standard-gage trucks with four or eight wheels, and equipped with a hoisting boom carrying a clam-shell or orange-peel bucket, shovel or electric magnet, depending upon the nature of the service to be ren-

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dered. The boiler and engine, or the operating motor, boom, hoisting drums, clutches, brakes, etc., are mounted on a turntable supported by the body of the crane—the engine or motor also actuating the propelling mechanism for moving the crane.

Locomotive cranes are ordinarily rated as of 10-, 15-, 20-, 25- or 30-ton lifting capacity, but such rating is misleading as it designates the ultimate lifting power of the crane with its boom elevated at a steep angle and a lifting radius of 12 or 13 ft. As the boom is lowered and the lifting radius increased, the capacity of the crane—its lifting power—drops rapidly, so that a small crane with a short lifting radius will handle a greater load than a considerably larger crane with a long lifting radius. Table 1 lists the lifting capacities of standard locomotive cranes in tons (2,000 lb.) for various lifting radii, from which the weight of the bucket, shovel or magnet should be deducted to arrive at the effective capacity of the crane. Table 2 gives the average weights of the standard sizes of clam-shell and orange-peel buckets ordinarily employed.

Table 1.—Lifting Capacity of Standard Locomotive Cranes

Lifting radius, feet	Size—rated capacity in tons				
	10	15	20	25	30
12–13	10.0	15.0	20.0	25.0	30.0
15	8.6	12.0	16.3	19.0	23.0
20	5.8	8.6	11.3	13.5	15.8
25	4.3	6.8	8.9	10.4	12.0
30	3.5	5.3	7.0	8.3	9.6
35	3.0	4.5	5.8	7.0	7.9
40	2.3	3.8	4.9	5.8	6.7
45	2.0	3.3	4.2	5.0	5.7
50	1.6	2.8	3.7	4.3	4.9
55	1.4	2.4	3.2	3.8	4.3
60	1.2	2.0	2.8	3.3	3.7

Table 2.—Average Weight of Standard Buckets

Type	Size, cubic yards	Weight, pounds	Type	Size, cubic yard	Weight, pounds
Clam-shell...	¾	2,200	Orange-peel..	¾	3,800
	1	2,800		1	4,800
	1½	3,800		1½	7,100
	2			2	9,500

Rarely is it possible to foretell the maximum proportional lifting radius at which a locomotive crane may be called upon to work, so considerable leeway in capacity must be allowed. The desired tonnage should be known, so a convenient empirical Formula (1) follows for ascertaining the size of crane best suited under ordinary conditions, while Formula (2) gives the average capacity of standard locomotive cranes in tons per hour when handling loose material.

Formula (1) $K = \frac{Q}{0.07w'}$ Formula (2) $Q = 0.07Kw'$

Where,

K = Rated capacity of locomotive crane in tons,
Q = Capacity in tons per hour (average) based on filling and discharging bucket 40 times per hour, and
w' = Weight of material handled in pounds per cubic foot.

Example 1.—Desired size of locomotive crane required to handle 125 tons of material weighing 80 lb. per cubic foot per hour.

$$K = \frac{125}{0 \cdot 07 \times 80} = 22.32. \quad \text{Use 20-ton locomotive crane.}$$

Example 2.—Desired average capacity of 15-ton locomotive crane when handling loose material weighing 80 lb. per cubic foot.

$$Q = 0 \cdot 07 \times 15 \times 80 = 84 \text{ tons per hour.}$$

Fuel Requirements.—Locomotive cranes are for the most part steam operated and on account of the intermittent nature of their operation average fuel consumption is the most reliable index of power requirements. With coal of average grade, the average fuel requirements for standard sizes of locomotive cranes are:

Table 3.—Fuel Requirements of Locomotive Cranes

10-ton crane.....	90 lb. per hour	20-ton crane.....	160 lb. per hour
15-ton crane.....	120 lb. per hour	25-ton crane.....	200 lb. per hour
	30-ton crane.....	240 lb. per hour	

Labor.—The crews customarily required to operate locomotive cranes are shown in Table 4.

Table 4

Size of crane, tons	Crew	
	Engineer	Helpers
10	1	1
15	1	1 or 2
20	1	2
25	1	2
30	1	2

Incidental supplies required for operating locomotive cranes constitute a charge per day of from 0.3 to 0.4 ct. per ton rating of crane per hour, such charge including the expenses contracted for oil, grease, waste, etc. Ordinary wear and tear (replacing cables, etc.), adds to the cost of operation an average of from 0.4 to 0.6 ct. per ton rating of crane per hour, making a total for incidental expenses of from 0.7 to 1.0 ct. per ton rating per hour operation. The initial cost of a locomotive crane varies from between $400 and $500 per ton of rated capacity, the smaller cranes being proportionally the more expensive. The burden (fixed charges)—represented by interest on investment, taxes, insurance, obsolescence, etc.—will constitute a yearly charge totaling to about 16.5 per cent of the initial cost of crane.

The net cost of operating standard sizes of locomotive cranes consists of the expenses for fuel, labor, incidental expenses and burden, or net cost per ton,

$$C = \frac{(F + L + E)T + B}{Q} \qquad \text{Formula (3)}$$

Where,

F = Fuel charge per hour (see Table 3 and price of coal),
L = Labor charge per hour (see Table 4 and labor rates),
E = Incidental expenses 0.7 to 1.0 ct. per ton rating per hour,
T = Task duration (yearly) in hours,
B = Burden 0.165 × ($400 to $500), and
Q = Capacity in tons per hour (see Formula (2)).

Example 1.—Required economic size and net operating cost (C) of locomotive crane for handling 50,000 gross tons (56,000 net tons) of loose material weighing 60 lb. per cubic foot within a period of 100 8-hr. days per year.

Size of crane, tons	Q, formula (2), tons	Task Duration	
		Hours	Days
10	42	1,334	Over 100
15	63	889	Over 100
20	84	667	83+
25	105	534	66+
30	126	444	55+

Assume.—Cost of fuel, $6 per ton (2,000 lb.) = $0.003 per pound,
Labor—engineer..... $1.00 per hour
Labor—helper..... $0.60 per hour
Incidental expenses, average—0.85 ct. per ton rating per hour;
Initial cost, $450 per ton rated capacity.
Twenty-ton locomotive crane—Formula (3):

$$C = \frac{(160 \times 0.003 + (1 + 1.2) + 20 \times 0.00085)667 + 0.165 \times 450 \times 20}{56{,}000}$$

= $0.0586—Net operating cost per ton.
Twenty-five-ton locomotive crane:

$$C = \frac{(200 \times 0.003 + (1 + 1.2) + 25 \times 0.00085)534 + 0.165 \times 450 \times 25}{56{,}000}$$

= $0.0600—Net operating cost per ton.
Thirty-ton locomotive crane:

$$C = \frac{(240 \times 0.003 + (1 + 1.2) + 30 \times 0.00085)444 + 0.165 \times 450 \times 30}{56{,}000}$$

= $0.0631—Net operating cost per ton.

Result.—Economic size, 20-ton locomotive crane, net operating cost, 5.86 cts. per ton.

Example 2.—Required net cost of handling 80,000 net tons of loose material weighing 102 lb. per cubic foot with a 25-ton locomotive crane and the duration of the task.

$Q = 0.07 \times 25 \times 120 = 210$ tons per hour—formula (2).
Task duration: 80,000 ÷ 210 = 381 hr., 47⅝ 8-hr. days.

Assume.—Cost of fuel, $6 per ton = $0.003 per pound.
Labor { engineer..... $1.00 per hour
helper..... $0.60 per hour
Incidental expenses, average—0.90 ct. per ton rating per hour.
Initial cost of crane, $400 per ton rated capacity.

By formula (3):

$$C = \frac{(200 \times 0.003 + (1 + 1.2) + 25 \times 0.0009)\, 381 + 0.165 \times 4000 \times 25}{80{,}000}$$

= $0.0341—Net operating cost per ton.

Result.—Net operating cost, 3.41 cts. per ton, duration of task, 48 8-hr. working days.

Hoists and Cranes.—Under this head may conveniently be grouped: (1) Simple mast and gaff rigs used for loading and unloading loose material by means of buckets, scoops, magnets, etc., which have a conveying radius controlled by the length of the gaff or boom of the rig. (2) Hoisting towers which are used primarily for unloading purposes, hoisting the material and transporting it to a greater or lesser extent according to whether the tower is of the traveling or stationary variety. (3) Overhead cranes which raise and lower loads, distributing them by means of trolley travel on the bridge and the movement of the bridge on its supporting rails. (4) Miscellaneous cranes such as those of the wall and jib types which have a more restricted area of operation and those special cranes developed for particular purposes.

Economic comparisons and detailed investigations into the cost of operating all these various types of hoists and cranes cannot well be made here, but an analytical consideration of the more complicated type, the overhead cranes, will serve to demonstrate approved methods for arriving at such data.

Overhead Cranes.—These are usually electrically operated, consisting of a hoist mounted on a traveling trolley running on a transverse, overhead bridge which in turn travels on supporting rails or tracks. The hoists employ buckets, tubs, magnets, slings, etc. for holding the material handled.

Overhead electric cranes are ordinarily rated as of 5-, 10-, 15-, 20-, 25-, 30-, 40-, 50-, 60-, 75-, 100- or 150-ton lifting capacity at its minimum, or first, hoisting speed. At higher hoisting speeds, its lifting capacity is reduced proportionally. Cranes are classified according to their hoisting speeds and are customarily classed as low-, medium- or high-speed. Each class of crane is provided with three hoisting speeds —the crane developing its full rated lifting capacity at first hoisting speed. At other than first speed, the lifting capacity is:

$$Q = \frac{K \times \text{first speed}}{\text{Operating speed}} \qquad \text{Formula (4)}$$

Where, K = rated capacity of crane in tons at first speed, and

Q = Lifting capacity of crane in tons at operating speed.

Hoisting speeds of overhead cranes are not standardized, but practice has fairly established speed ratios approximately as given in Table 5. Trolley speeds vary from 100 to 150 ft. per minute. Bridge speeds vary from 300 to 500 ft. per minute for small cranes and from 200 to 300 ft. per minute for large cranes.

Power.—Power requirements of overhead cranes for hoisting, trolley and bridge motors depend upon speed of travel (minimum) and the load lifted (maximum capacity). The power of the motors are thus subject to selection to a certain extent and is governed by the Formulas (5*a*), (5*b*) and (5*c*):

$$\text{Horsepower of hoisting motor} = \frac{Q \times HS}{16} \qquad \text{Formula (5}a\text{)}$$

$$\text{Horsepower of trolley motor} = \frac{Q \times TS}{400} \qquad \text{Formula (5}b\text{)}$$

$$\text{Horsepower of bridge motor} = \frac{(Q + 0.03L)BS}{235} \qquad \text{Formula (5}c\text{)}$$

TABLE 5.—HOISTING SPEEDS OF STANDARD OVERHEAD ELECTRIC CRANES

Rated capacity in tons	Speeds in feet per minute								
	Class								
	Low			Medium			High		
	First	Second	Third	First	Second	Third	First	Second	Third
5	27	34	40	47	60	70	66	83	100
10	20	24	30	37	48	57	50	62	75
15	16	20	24	29	36	42	40	50	60
20	14	17	20	24	28	34	32	40	48
25	12	15	18	20	24	28	28	34	40
30	11	14	16	18	21	24	24	29	35
40	9	11	14	16	18	21	21	25	30
50	8	10	12	14	16	19	19	22	26
60	7	9	11	13	15	18	18	20	24
75	6	8	10	12	14	17	17	19	22
100	5	6	9	10	12	15	15	17	20
150	4	5	7	8	10	12	12	15	18

Where, Q = rated capacity of crane in tons,
HS = minimum, or first, hoisting speed in feet per minute,
TS = trolley speed in feet per minute,
BS = bridge speed in feet per minute, and
L = span of bridge in feet.

TABLE 6.—COMPARATIVE COSTS OF STANDARD OVERHEAD ELECTRIC CRANES

Rated capacity	Span in feet							
	20	25	30	35	40	50	75	100
5	0.80	1.00	1.03	1.07	1.10	1.20	1.40	1.70
10	1.20	1.25	1.30	1.35	1.40	1.50	1.80	2.15
15	1.40	1.45	1.51	1.57	1.62	1.75	2.09	2.54
20	1.59	1.64	1.70	1.77	1.83	1.98	2.35	2.88
25	1.65	1.80	1.88	1.94	2.01	2.17	2.61	3.19
30	1.90	1.97	2.04	2.10	2.19	2.38	2.88	3.50
50	....	2.50	2.60	2.70	2.80	3.00	3.52	4.18
75	....	3.10	3.20	3.35	3.41	3.68	4.32	5.18
100	....	3.60	3.71	3.85	3.99	4.25	5.00	....
150	....	4.28	4.40	4.52	4.67	4.98		

Cost.—The cost of overhead electric cranes varies between quite wide limits, but reliable approximations can be arrived at by comparative costs based on a known cost of some specific crane of standard type and given span. Index values based on

the average cost of a 5-ton, medium-speed, crane of 25-ft. span taken as 1 are given in Table 6. This data provides sufficiently reliable values for comparing the cost of various sizes of cranes and for purposes of estimating the probable cost of the equipment, etc.

Base: 5-ton, medium-speed, crane of 25-ft. span = 1.

Depreciation of overhead electric cranes is customarily figured at 5 per cent of cost—a safe value—while ordinary repairs should not average more than 2 per cent. Interest on investment, insurance, taxes, etc., add another 8 per cent of the initial cost of the equipment per year, making the average burden 15 per cent. One man to run the crane with some occasional service from an assistant constitutes the necessary operating crew, the expense of which should not average per hour more than one and three-seventh times the hour-wage rate of the craneman. Oil, waste and incidental supplies, including the expense of an occasional careful overhauling and thorough cleaning of equipment, varies closely with the total power consumption of the crane and may be taken at a mean value of 0.15 ct. per total motor horsepower per hour. The average power requirements, including the power required to run the crane empty as well as under load is in kilowatts per hour taken as equal to half the total horsepower requirements of the crane expressed in horsepower per hour.

The total net cost of operating the crane for a period of a year should be the basis of estimating the net operating cost per hour or per ton handled, in order that the burden expense may be equitably apportioned. If this is not done, it is necessary to allot arbitrarily a certain proportion of the yearly burden to the operating expense for the period under consideration. The net cost of operating a standard overhead electric crane, expressed in the cost per ton handled and per hour of operation, follow as Formulas (6*a*) and (6*b*):

$$C/t = \frac{(P + L + E)T + B}{Q'} \quad \text{Formula (6}a\text{)} \qquad C/hr = C/t \times Q'' \quad \text{Formula (6}b\text{)}$$

Where,

P = Average power cost = consumption in kilowatts per hour times unit rate = 0.5 times total crane horsepower per hour times unit power rate,

L = Labor charge per hour = 1.43 times hour-wage rate of craneman,

E = Incidental expenses per hour = 0.15 times total crane horsepower per hour,

T = Task duration (yearly) in hours,

B = Burden (yearly) = 0.15 times initial cost of equipment,

Q' = Quantity of material handled in tons, and

Q'' = Quantity of material handled in tons per hour—bucket, or crane, with average load.

An overhead electric crane is equipment which is rarely in continuous operation, so other conditions than simply its lifting capacity govern, as a rule, its selection. The maximum load to be handled must be known, total tonnage to be handled per year estimated and the number of probable operating hours established. The service required may be of a special nature, such as handling a heavy load once, or a few times per year, in which case the lifting capacity of the crane would be the one governing factor. Or, a quantity of product or other commodity measured in tons per year might have to be handled, complicating the selection of the economic size of crane to install. The selection in such case, though allowance for expansion should always be made, entails a knowledge of all governing factors and is best illustrated by considering a typical example—setting down in detail the steps customarily taken.

Example 3.—Required economic size and net operating cost of overhead electric crane to handle 120,000 tons of material per year (300 8-hr. working days), operating an average of 2 hr. per day. Specifications: 50 ft. span, 40 ft. lift (maximum), 240 ft. bridge travel (maximum); medium-speed class crane.

T (minutes) = 300 × 2 = 600 hr.
Average capacity per hour = 120,000 ÷ 600 = 200 ton.
Assume average load = two-thirds full load.
Required capacity per hour = 200 ÷ ⅔ = 300 ton.

Assume bridge speed = 400 ft. per minute,
Average bridge travel (loaded) per trip = ⅔ × 240 = 160 ft.
Travel per trip (loaded and empty) = 160 × 2 = 320 ft.

Duration of bridge travel per trip = 320 ÷ 400 = 0.80 min.
Assume average dumping time = 1.00
Loading time = 1.20
3.00

Assume trolley speed = 120 ft. per minute,
average trolley travel per trip = 50 ft.
Time required for trolley travel per trip = 50 ÷ 120 = 0.42 min.
(Note: Time required is less than that of bridge travel, so may be ignored.)
Disregarding hoisting time, 20 trips (60 ÷ 3) per hour could be made. In which case, K = 300 ÷ 20 = 15 ton.
HS (hoisting speed) for 15-ton, medium-speed, crane (first speed) = 29 ft. per minute (Table 5).
Maximum lift = 40 ft., requiring 1⅔ min. (40 ÷ 29)
1.20 + 0.5 × 0.80 = 1.60 min. available.
This does not allow sufficient margin, so try K = 20.

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300 ton (required capacity per hour) ÷ 20 = 15 trips per hour required. Allows 4 min. per trip.
HS for 20-ton, medium-speed, crane (first speed) = 24 ft. per minute.
Maximum lift = 40 ft., requiring 1.66 min. (40 ÷ 24)
More than 2.00 min. available.

Select 20-ton crane, medium speed; hoisting speed (first), 24 ft. per minute; trolley speed, 120 ft. per minute; bridge speed, 400 ft. per minute.
As average load is taken as two-thirds full load, Q'' = 20 ton.

Power requirements:

$$\text{Hoisting motor} = \frac{20 \times 24}{16} = 30 \text{ hp.} \qquad (5a)$$

$$\text{Trolley motor} = \frac{20 \times 120}{400} = 6\text{—say } 10 \text{ hp.} \qquad (5b)$$

$$\text{Bridge motor} = \frac{(20 + 0.03 \times 50)400}{235} = 36.6\text{—say } 40 \text{ hp.} \qquad (5c)$$

Total ..... 80 hp.

Initial Cost.—Take cost of 5-ton, medium-speed, 25-ft. span, crane = $5,000. Cost of 20-ton, medium-speed, 50-ft. span, crane = 5,000 × 1.98 = 9,900—say $10,000 (Table 6).

Unit rates (assumed):

Craneman..... $1.00 per hour

Current..... 0.05 per kilowatt-hour.

Net operating cost:

$$C/t = \frac{(0.5 \times 80 \times 0.05 + 1.43 \times 1 + 0.15 \times 80)\ 600 + 0.15 \times 10{,}000}{120{,}000}$$

$$= \frac{10{,}758}{120{,}000} = \$0.0896 \qquad (6a)$$

$$C/\text{hour} = 0.0896 \times 300 = 26.88\text{—say } \$27 \text{ per hour} \qquad (6b)$$

Miscellaneous Cranes.—Wall and jib cranes, whether stationary or of the traveling type, hoists of various kinds and all other varieties of cranes lend themselves to economic selection and analyses as to probable net cost of operation along lines very similar to those followed in the case of overhead electric cranes. A full knowledge of operating requirements is necessary, suitable provisions should be made for possible expansion, and the mistake avoided of assuming too great a mechanical efficiency for the equipment. An intimate knowledge of costs—labor and equipment—and depreciation expenses is also required for even such approximate estimates, if they are to be at all reliable.

CONVEYORS AND ELEVATORS

Classification.—The classification and terminology for conveying and elevating equipment has grown quite confused and as conveyors are frequently installed at an inclination so that they elevate as well as convey, it is well to classify such material handling equipment as carries a continuous, or approximately so, load under the one head of conveyors and elevators, to distinguish them sharply from other material handling equipment which handle more intermittent loads.

The distinctive merits possessed by conveyors and elevators as thus defined are large hourly capacity, low power consumption and relatively light construction. They are used chiefly to handle loose material in bulk, though special designs are employed for handling material in containers, usually of similar dimensions and weight.

Belt Conveyors.—Endless belts running about end pulleys and supported on the runs by a series of securely mounted idlers, usually of the troughed variety on the conveying run and flat on the return run, though "special purpose" belt conveyors are sometimes furnished with straight, or flat, idlers on the carrying run. The individual pulleys comprising the troughed type of idlers are of uniform diameter so that the peripheral speed of the pulleys at all points of contact with the traveling belt is the same as the belt speed. Troughing idlers may consist of any number of individual pulleys, but an odd number is to be preferred, in order that the center section of the conveying belt may be properly supported (see Fig. 1).

Return idlers, supporting the return run of the belt, are flat, made up of a series of individual pulleys mounted on a common shaft or a single pulley slightly wider than the width of the belt it supports. These return idlers are spaced at about 10 ft. intervals, while the troughing and load-carrying idlers are spaced from 2 to 5 ft. apart, depending upon the width of the belt and the weight of the material handled—the wider the belt, the closer are the idlers spaced. At all loading points, the troughing idlers should be spaced at about two-thirds the spacing required on conveying stretches

of the belt and a troughing idler should be placed about 6 in. to the rear of the discharging lip of the loading chute to give added support to the belt. Where the carrying belt takes an abrupt change in direction, a straight-face bend pulley is substituted for a troughing idler at the point at which the bend occurs and the troughing idlers immediately preceding and following the bend pulley should be situated from it at distances somewhat less than the spacing of idlers on straight carrying sections of the conveyor. When belt conveyors exceed 25 ft. in length—they have been built up to 1,000 ft. in length—guide idlers mounted normally to the edges of the belt on the carrying side should be installed about every 25 ft. to prevent the belt from leaving the troughing idlers.

Discharge of load occurs over the head-end pulley of the conveyor or may take place at any point along the conveyor if the carrying belt is carried over an elevated

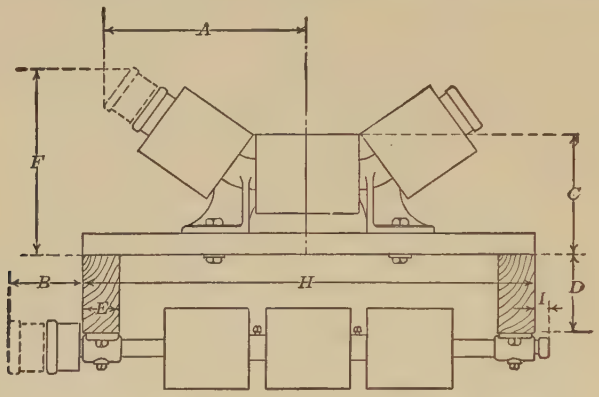


Fig. 1.—Standard three pulley belt-conveyor idlers.

discharge pulley (straight) and then around a lower bend pulley (straight) situated slightly to the rear of the discharge pulley so that the carrying belt takes a reverse S-loop. Such discharge points may be permanently located, in which case they are termed "fixed dumps," or the discharge and bend pulleys may be mounted on a traveling carriage and shifted from place to place on a horizontal section of the conveyor, in which case the discharging carriage is termed a "tripper." Trippers may be of the hand-propelled type or the power for their movement can be secured from the traveling belt—automatic traveling trippers. By a system of stops, automatic trippers may be made to travel automatically back and forth between designated points, distributing the belt load uniformly over their discharging path.

Loading and discharging belt conveyors should be through the agency of chutes so inclined that the velocity of the material leaving the chute should be as nearly as possible the same as the speed of the conveyor belt, and in the same direction.

The driving pulley, which should be heavily crowned in order to maintain alignment of belt, is advisably the head pulley of the conveyor, in order that the tension side of the belt may carry the load, but when this arrangement is not feasible the drive may be located at the tail end of the conveyor or at any poin along its lower run by looping the return belt much as the carrying belt is looped for a discharge point. For unusually long belt conveyors, two driving pulleys geared together are employed, in which case the driving section of the belt forms a double loop. A still more powerful drive is secured by covering the driving pulleys with lagging to increase their grip on the conveyor belt. Typical types of conveyor drives are shown in Fig. 2.

Owing to the unavoidable stretch of the conveyor belt, particularly when first installed and operated, take-ups with generous adjustment should be provided, customarily at the end of the conveyor removed from the drive. Take-ups can also be installed on the return belt on a loop similar to that required for center drives.

If the conveyor is to be employed for handling material which is damp or which has any tendency to adhere to the belt, rotary brushes should be installed at the head end

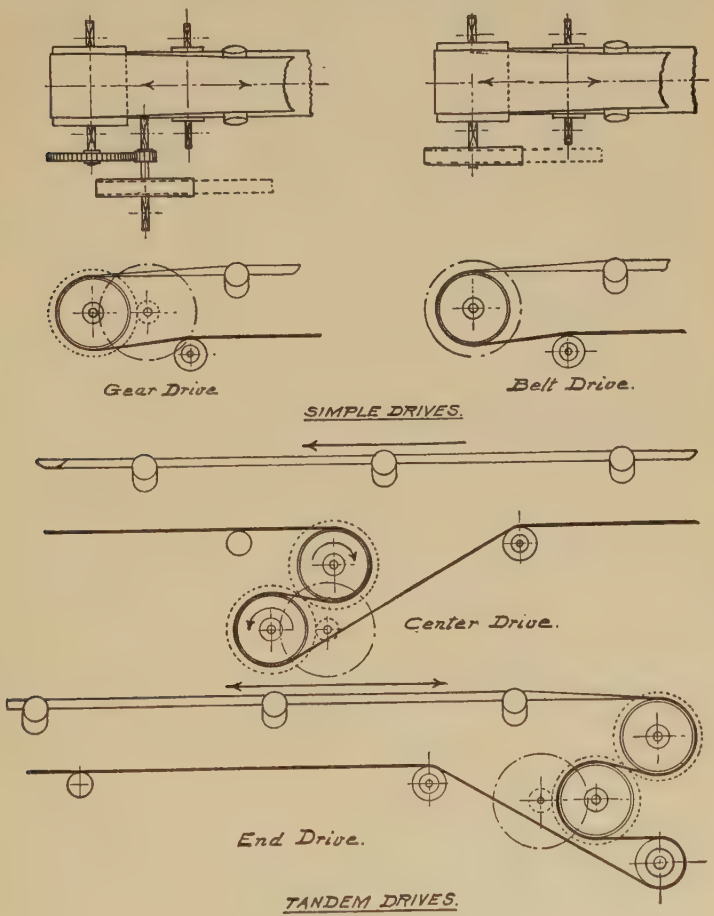
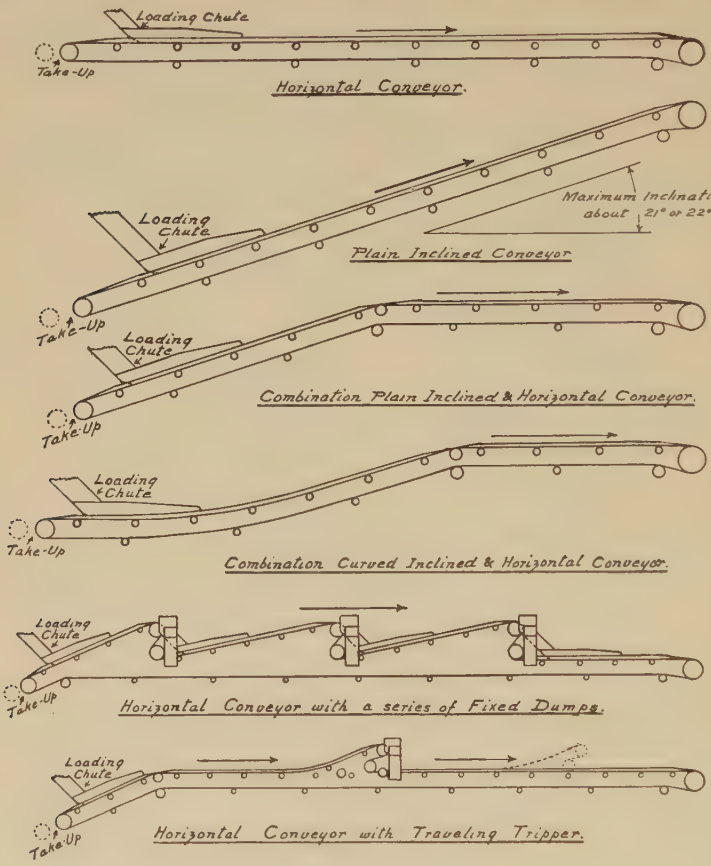


FIG. 2.—Typical belt-conveyor drives.

of the conveyor and at the discharge pulleys of fixed dumps and trippers. The brush should be set against the under side of the belt a little to the rear of the conveyor pulley, so that the trajectory of the sweepings will cause them to land on the discharging chutes. The rotation of the brush is secured from that of the conveyor pulley and should be considerably more rapid than that of the conveyor pulley—the brush sweeping against the travel of the return run of conveyor belt, *i.e.*, rotating in the same direction as the conveyor travel.

Loading chutes should not only be of generous pitch to secure the desirable speed of material flow but should discharge to the conveyor belt over a curved lip so as to

minimize the shock of impact and should have guiding skirt boards spanning the space from two to three troughing idlers to allow the load to settle and adjust itself on the belt before it passes the confines of the skirt boards. Triangular cleats should run



Notes:—
Installations of Belt Conveyors may consist of any combination of the above arrangements and conveyors can frequently be made reversible—i.e. arranged so they can be operated in either direction. They may also be loaded at more than one point, can be driven at any point and the Take-Up can be located wherever convenient.
Fixed Dumps may be installed any horizontal stretch of conveyor or inclined stretches may be interrupted for their accommodation.
Traveling Trippers may be installed on any horizontal or slightly inclined stretches of conveyor, may be propelled by hand or be of the automatically traveling and reversing type.

FIG. 3.—Typical belt-conveyor arrangements.

the full length of the skirt boards to assist in properly distributing the load and the distance between the skirt board should be several inches less than the width of the conveyor belt, depending upon the width of the belt.

Speeds of belt conveyors vary greatly from 40 to 50 ft. per minute for picking and sorting conveyors to several hundred feet. Heavy material, as a rule, can be conveyed at a higher speed than light material.

Belt conveyors are limited in their ability to carry loads up an incline of much more than 20 deg. and have to be erected in one vertical plane—*i.e.*, cannot turn corners—though a series of conveyors can discharge from one to the other, each running in a

different direction. Despite these limitations, they lend themselves to quite a variety of arrangements (see Fig. 3). The path of conveyors over curved inclines should be a catenary, but is quite generally made a circular arc of about 250 ft. radius. Installations of movable trippers and the first fixed dump require that the tail pulley of the conveyor be sufficiently removed that when the discharging device is at the limit of its backward travel the conveyor belt will not lift from the troughing idlers in the vicinity of the tail pulley.

Table 7.—Economic Speeds of Belt Conveyors Handling Various Materials

Material	Average weight pounds per cubic foot	Belt speed feet per minute
Coke	33.5	250
Broken stone (coarse)	165.0	275
Lump coal	55.0	275
Ashes	45.0	300
Lime and cement	65.0	300
Ore (average)	125.0	350
Crushed stone	160.0	375
Sand and gravel	110.0	375
Fine coal	50.0	400

The inclination of the pulleys of the troughing idlers turning up the edges of the belt affect the capacity of the conveyor, but as the capacity is so increased the power requirements increase more rapidly and also the wear on the conveyor belt. For this reason the degree of troughing has become pretty well standardized at an economic point with the end troughing pulleys turning up the belt to from 20 to 25 deg. With the degree of troughing so established, the capacity of the conveyor is controlled by the weight of the material handled and the belt speed. Table 7 lists the economic speeds for certain classes of material customarily handled on belt conveyors and Table 8 gives the average capacity of standard sizes of conveyors when continuously and uniformly loaded with material weighing 100 lb. per cubic foot, while Formula (7) gives the equation for calculating the capacity of troughed belt conveyors handling material of known weight at any given speed.

$$W = \frac{Kw^2Vw'}{100{,}000} \qquad \text{Formula (7)}$$

W = Capacity in tons per hour.

Table 8.—Capacity of Belt Conveyors at 100 Ft. per Minute

Material weighing 100 lb. per cubic foot

Width of belt, inches	Tons per hour	Width of belt, inches	Tons per hour
12	20.0	26	96.0
14	27.0	28	108.6
16	35.7	30	126.8
18	45.0	32	143.5
20	55.6	34	160.0
22	67.7	36	183.0
24	80.0	42	265.0

K = Constant = 1.43 to 1.65—average, 1.5,
w^2 = Width of belt in inches squared,
V = Speed of conveyor in feet per minute,
w' = Weight of material handling in pounds per cubic foot.

Power Requirements.—The consumption of power by belt conveyors depends somewhat upon the method of their lubrication, but is so low that the power requirements for well grease-lubricated equipment may be taken as the standard and reduced by an arbitrary amount to be approximately accurate for equipment mounted on ball or roller bearings. Formula (8) gives the power requirements for a grease-lubricated conveyor well aligned and properly cared for.

$$\text{Horsepower} = \frac{(0.00009w^2V + 0.08W)L + WH}{1{,}000} \qquad \text{Formula (8)}$$

Where,

w = Width of conveyor belt in inches,
V = Speed of conveyor in feet per minute,
W = Load handled in tons per hour,
L = Total length of conveyor in feet, and
H = Rise in length of conveyor in feet.

Horsepower additional required for each tripper or fixed dump = 1 hp. for each inch of belt width in excess of 10 in.

For ball- or roller-bearing conveyors reduce power requirements on horizontal stretches by 33⅓ per cent.

The costs of belt conveyors and of conveyors and elevators in general are so susceptible to the market prices for materials and to manufacturing costs that exact figures cannot well be presented, but relative costs can be arrived at which will enable accurate estimates of market prices to be arrived at by comparison and which will serve as reliable guides for ascertaining the relative economy of various types of conveyors and elevators. This method of presenting costs will be adhered to for all types of conveyors and elevators discussed, the relative costs established on a common base so as to be mutually comparable.

Formula (9) gives the *relative* cost of high-grade belt conveyors fully equipped with durable rubber belt and all accessories; Formulas (9*a*), (9*b*) and (9*c*), the relative costs of discharging devices; while Formula (10) gives an equation for ascertaining the annual depreciation—expressed in the same relative units—to be anticipated with proper care and attention to the equipment.

$$RC_{bc} = (0.45w - 1.7)L + 3.65w \qquad \text{Formula (9)}$$
$$RC_{fd} = 2w + 12 \qquad \text{Formula (9}a\text{)}$$
$$RC_{ht} = 6.3w + 75 + 0.81L_t \qquad \text{Formula (9}b\text{)}$$
$$RC_{at} = 11w + 132.5 + 0.81L_t \qquad \text{Formula (9}c\text{)}$$
$$RD_{bc} = (0.08w - 0.32)L + 0.365w \qquad \text{Formula (10)}$$

Where,

RC_{bc} = Average relative cost of belt conveyors,
RC_{fd} = Average relative cost of fixed dumps,
RC_{ht} = Average relative cost of hand propelled trippers,
RC_{at} = Average relative cost of automatic trippers,
RD_{bc} = Average relative depreciation—annual,
w = Width of conveyor belt in inches,
L = Total length of conveyor in feet, and
L_t = Lengthtripper travel in feet.

Fixed charges, consisting of interest on investment, insurance and taxes, etc., amount to about 8½ per cent of the initial cost of equipment.

Labor expense, almost a negligible factor with belt conveyors, may be arbitrarily set at 1½ cts. per inch of width of conveyor per hour, irrespective of the load handled. Incidental expenses for supplies to keep the equipment in good operating condition vary closely with the power requirements and may be taken at ¾ ct. per horsepower per hour.

The total expense of operating a belt conveyor system is best expressed in cost per ton of material handled and this is best illustrated by a specific example.

Example.—Required the net operating cost per ton of a 400-ft. conveyor handling 60,000 tons of fine coal during 1,200 operating hours in the year. The conveyor to elevate the coal 24 ft. and distribute it by means of an automatic tripper over a bunker 60 ft. long. Power delivered at the conveyor drive commands a value of 3 cts. per horsepower.

Advisable speed of conveyor (Table 7) = 400 ft. per minute.

Minimum capacity of conveyor (60,000/1,200) = 50 tons per hour.

Minimum capacity of conveyor at 100 ft. per minute, 12.5 tons per hour.

Fine coal weighs 50 lb. per cubic foot, so required size of conveyor (Table 8) equals 14 in.

Power requirements:

$$\text{Horsepower} = \frac{(0.00009 \times 14^2 \times 400 + 0.08 \times 50)\ 400 + 50 \times 24}{1{,}000} \qquad \text{Formula (8)}$$

5.62—say 6 hp.

Additional horsepower for tripper..... 4 $(14 - 10)$

Total.....10

Costs:

$$RC_{bc} = (0.45 \times 14 - 1.7)400 + 3.65 \times 14 = 1{,}291.10 \qquad \text{Formula (9)}$$

$$RC_{at} = 11 \times 14 + 132.5 + 0.81 \times 60 = 335.10 \qquad \text{Formula (9c)}$$

Total relative cost = 1,291.10 + 335.10 = 1,626.20.

Take market price at 25 per cent above relative costs.

Cost = 1.25 × 1,626.20 = 2,032.75—say $2,000.00

Fixed charges:	(0.085 × 2,000).....	$170.00
Labor charge:	(0.015 × 14)1,200.....	252.00
Supply expense:	(0.0075 × 10)1,200.....	90.00
Power charge:	(0.03 × 10)1,200.....	360.00
Depreciation expense:	$1.25 \times RD_{bc}$	
	$=1.25(0.08 \times 14 - 0.32)400 + 0.365 \times 14 = 51.10$ say	50.00
		$922.00
Net operating cost:	(922/60,000).....	$0.0154 per ton

Flight conveyors are endless chains (one or two) carrying attached flights and dragging the load through troughs. Flights may rest on the inclined sides of the trough, be suspended from supporting and wearing shoes (shoe flight conveyors) or may be supported by flanged rollers running on rails (roller flight conveyors).

Flights, unless supported on the inclined sides of the trough, are usually rectangular, with their length two and one-half to two times their width, and equally spaced from 12 to 24 in. Sheet-steel flights are customarily used, unless the service is so exacting as to require cast or malleable iron. Troughs are usually steel lined and clearance should be provided between the bottom edge of the flights and the trough bottom for the collection of a protecting covering of fines.

Discharge of load takes place on the lower run of the conveyor through sliding gates set in the trough. Both long- and short-link chains are employed, the former having the advantage of presenting fewer bearing surfaces (fewer joints, or links), but produce an intermittent tractive effort in passing about the polygonal end sprockets. The advantages and disadvantages in the case of short-link chains are simply reversed. The sprockets in either case are advisably provided with chilled teeth to resist excessive wear.

Conveyor drives should be set in rigid frames with reductions through gearing or by means of silent chains and lubrication should be with a high-grade of medium-body cup grease liberally supplied. Speeds vary from 100 to 200 ft. per minute depending upon the load. The carrying capacity of flight conveyors varies directly with the effective area of the flights, speed of conveyor and weight of the material and indirectly with the spacing of the sights. Inclining a flight conveyor decreases its carrying capacity from 1⅔ to 2 per cent for each degree of angularity. As most flight conveyors are equipped with flights of rectangular form, it will suffice to consider only such conveyors. As the spacing of flights is pretty well standardized at 12, 16, 18 or 24 in. and the weight of load is a fixed quantity, the potent factor in capacity, other than size of flights and their spacing, is the speed at which the conveyor is run. This varies with the character of the material handled and Table 9 lists the economic speeds for various materials.

Table 9.—Economic Speeds for Flight Conveyors Handling Various Materials

Material	Speed, Feet per Minute
Coke	100
Broken stone (coarse)	125
Lump coal, run-of-mine	125
Ashes	150
Lime and cement	150
Ore (average)	175
Crushed stone	175
Sand and gravel	175
Fine coal	200

Formulas for ascertaining the capacity of flight conveyors with rectangular flights equally spaced and the conveyors economically and continuously loaded are:

$$W_h = \frac{0.0028AVw'}{S} \qquad \text{Formula (11)}$$

Where,

W_h = Capacity in tons per hour—horizontal conveyor,
A = Area of flights in square inches,
w' = Weight of material handled in pounds per cubic foot, and
S = Spacing of flights in inches.

Power Requirements.—The consumption of power in operating flight conveyors is materially affected by whether the flights are mounted on sliding shoe or roller flights. Formula (12) by the use of the proper constant permits the calculation of power requirements in either event.

$$\text{Horsepower} = \frac{CWL + WH}{1{,}000} \qquad \text{Formula (12)}$$

Where,

C = Constant = 0.780 for roller flights,
= 0.933 for shoe flights,
W = Load handled in tons per hour,
L = Length of conveyor in feet, and
H = Elevation in length of conveyor in feet.

The multiplicity of standards and the differing spacing of flights makes an accurate formula for ascertaining cost a difficult matter, but for all practical purposes for approximate estimates of relative cost Formula (13) will be found sufficiently reliable.

$$RC_{fc} = (0.0115A + 0.0516\sqrt{A} + 7)L + K\sqrt{A} \quad \text{Formula (13)}$$

Where,

A = Area of flights in square inches—width times length,
L = Length of conveyor in feet, and
K = Constant = 1.5 for shoe flights,
= 3.0 for roller flights.

Depreciation varies greatly in the case of flight conveyors as both the flights and the trough are subject to severe abrasive action by the material with which they are in contact in operation. Virtually every material varies in its destructive action. With Formula (14), depreciation factors for some of the materials commonly handled by flight conveyors are given.

$$RD_{fc} = ((0.00323A + 0.00237\sqrt{A} + 1.075)L + 0.6\sqrt{A})F \quad \text{Formula (14)}$$

Where,

F = Depreciation factor = 0.47 for coke
0.99 for ashes
1.00 for fine coal
1.06 for lump coal, run-of-mine
1.15 for lime and cement
1.50 for sand and gravel
1.60 for ore (average)
1.80 for crushed stone
1.83 for broken stone (coarse)

Fixed charges for flight conveyors may be taken, as in the case of belt conveyors, at 8.5 per cent per year, but in addition there is an annual renewal charge amounting to about 20 per cent of the initial cost.

Labor expense, though no higher per ton than that of belt conveyors, is considerably more on account of the necessary attention to gates, etc., per foot of conveyor and is usually about 3 cts. per inch width of conveyor, irrespective of the load handled or the length of the conveyor. Flight conveyors owing to their heavy consumption of power are much more limited in length than are belt conveyors. Incidental expenses for supplies, etc., vary closely with the power requirements and are somewhat heavier in the case of shoe-flight conveyors, being for roller-flight conveyors about 2 cts. per horsepower per hour and for shoe flights about 2.25 cts. per horsepower per hour.

The respective net costs of operating the two types of flight conveyors differ to some extent, as will be noted in the following example.

Example.—Required the net operating cost per ton of a 100-ft. flight conveyor handling 120,000 tons of sand and gravel during 800 operating hours in the year. The conveyor to be inclined about 3 deg. (5.25 ft. in its length). Power to command a value of 3 cts. per horsepower delivered at conveyor drive. Costs wanted for both shoe- and roller-flight conveyors.

Advisable speed of conveyor (Table 9)..... 175 ft. per minute
Minimum capacity of horizontal conveyor (120,000/800) 150 tons per hour
Required minimum due to inclination (150/0.97)..... 155 tons per hour
Assume spacing of flights as 18 in., and weight of sand and gravel at.....110 lb. per cubic foot

$$W_h = 150 = \frac{0.0028 \times 175 \times 110 \times A}{18} \qquad \text{Formula (11)}$$

A = 50 sq. in. + 6 per cent for inclination = 53 sq. in.

Take A = 60 for safety and flights 5 × 12 in.

Power requirements:

$$\text{Horsepower} = \frac{0.780 \times 150 \times 100 + 150 \times 5.25}{1{,}000} = 12.49\text{—say 15 hp.}$$

$$\text{Horsepower} = \frac{0.933 \times 150 \times 100 + 150 \times 5.25}{1{,}000} = 14.78\text{—say 18 hp.} \qquad \text{Formula (12)}$$

Costs:

$$RC_{fc} = (0.0115 \times 60 + 0.0516 \times \sqrt{60} + 7)100 + 1.5\sqrt{60}$$
$$= 1{,}441.63\text{—say 1,450, for shoe flights}$$
$$= (0.0115 \times 60 + 0.0516 \times \sqrt{60} + 7)100 + 3.0\sqrt{60}$$
$$= 1{,}153.25\text{—say 1,465, for roller flights} \qquad \text{Formula (13)}$$

$$RD_{fc} = ((0.00323 \times 60 + 0.00237\sqrt{60} + 1.075)100 + 0.6\sqrt{60})1.50$$
$$200.05\text{—say 200.} \qquad \text{Formula (14)}$$

Take market price at 25 per cent above relative costs.

Cost = 1.25 × 1,450 = 1,812.50—say $1,815.00—shoe flights
= 1.25 × 1,465 = 1,831.25—say $1,835.00—roller flights.

		Shoe Flights	Roller Flights
Fixed charges:	(0.085 × 1,815).....	$154.28	
	(0.085 × 1,835).....	.....	$155.98
Labor charge:	(0.03 × 12)800.....	288.00	288.00
Supply expense:	(0.03 × 18)800	432.00	
	(0.02 × 18)800.....	.....	288.00
Power charge:	(0.03 × 18)800.....	432.00	
	(0.03 × 15)800.....	.....	360.00
Depreciation:	(1.25 × 200)	250.00	250.00
Renewal charge:	(0.20 × 1,815).....	363.00	
	(0.20 × 1,835).....	.....	367.00
		$1,919.28	$1,708.98

Net Operating Cost:

(1,919.28/120,000).....$0.016 per ton—shoe flights.
(1,708.98/120,000).....$0.01424 per ton—roller flights.

Screw conveyors are helical flights wound about a central shaft (screws) revolving in troughs or boxes through which the load is pushed by the screw. The size of the conveyor is designated by the diameter of the screw, but the diameter of the trough may vary between wide limits, from one which is only lsightly larger than the screw to a commodious box encasing the screw in which the material handled collects and the screw makes its own trough in a bed of material.

Screws for ordinary service customarily consist of steel ribbons wound helically around the central steel shaft or pipe, but for severe service and handling heavy loads the flights are made much heavier, usually of steel, but occasionally of cast iron. The screws are supported by a series of bearings spaced from 6 to 10 ft. centers and the troughs are closed at the charging end. The standard pitch is approximately equal to the diameter of the screw.

Load is discharged through the open far end of the conveyor or along the conveyor through bottom gates in the trough. The drive is customarily located at the charging end of the conveyor, but when this is not feasible it may be located at the discharge end.

Speeds of screw conveyors are not governed so much by the character of the load they handle as by their diameter and it has become customary to consider speeds approximating those given in Table 10 as standard for almost all materials handled by screw conveyors.

TABLE 10.—ECONOMIC SPEEDS FOR SCREW CONVEYORS

Diameter screw, inches	Revolutions per minute	Diameter screw, inches	Revolutions per minute
3	240	10	160
4	220	12	150
5	205	14	140
6	195	16	130
7	185	18	120
8	175	20	115
9	170	24	100

Capacity.—A general formula for ascertaining the capacity of standard screw conveyors handling various materials of known weight is given as Formula 15.

Where,

$$W = \frac{0.3231d^2w'R}{100{,}000} \qquad \text{Formula (15)}$$

W = Capacity in tons per hour,
d = Diameter of screw in inches,
R = Revolutions of screw per minute, and
w' = Weight of material handled in pounds per cubic foot.

The consumption of power by screw conveyors is relatively high, but they are usually short—a series of conveyors discharging from one to the other being used if the distance the material has to be handled is more appreciable—and have quite limited capacity, so the power consideration is not of great importance. The additional power required to lift the load should the conveyor be installed at an inclination is consequently not so marked. The inclination is limited to 10 or 15 deg. with the horizontal. A convenient formula for ascertaining power requirements of screw conveyors follows.

$$\text{Horsepower} = \frac{0.68WL + WH}{1{,}000} \qquad \text{Formula (16)}$$

Where,

W = Load handled in tons per hour,
L = Length of conveyor in feet, and
H = Elevation in length of conveyor in feet.

A simple equation for ascertaining the *relative cost* of common sizes of screw conveyors is given as Formula 17, but the one for calculating the *relative depreciation* per year is complicated by the introduction of the depreciation factor necessarily entailed when the moving parts of the conveyor are in direct contact with the load in a manner which produces a destructive abrasive, or scouring, action.

$$RC_{sc} = 0.196dL + 2.333d + 0.07d^2 \qquad \text{Formula (17)}$$

Where,

RC_{sc} = Relative cost of screw conveyor,
d = Diameter of screw in inches, and
L = Length of conveyor in feet.

$$RD_{sc} = (0.061dL + 0.25d + 0.007d^2)F \qquad \text{Formula (18)}$$

Where,

RD_{sc} = Relative depreciation (annual) of screw conveyor, and
F = Depreciation factor = 0.47 for coke; 1.00 for fine coal; 0.99 for ashes; 1.15 for lime and cement

Fixed charges for screw conveyors may be taken as 8.5 per cent a year by allowing an annual renewal charge of 20 per cent of initial cost. Labor expense of about ¾ ct. per inch diameter of screw during hours of operation and a supply expense of 1⅛ cts. per horsepower constitute the other charges included in the net cost of operation.

The total cost of operating a screw conveyor is best indicated by the consideration of a typical example, as follows:

Example.—Required the net operating cost per ton of a horizontal screw conveyor, 12-in. diameter and 24 ft. long, handling 10,000 tons of cement—weighing 65 lb. per cubic foot—over a period of a year. Power delivered at conveyor drive commanding a value of 3 cts. per horsepower.

Advisable speed of conveyor (Table 10) 150 r.p.m.

$$\text{Capacity, } W = \frac{0.3213 \times 12^3 \times 65 \times 150}{100{,}000} = 12.81\text{—say 10 tons per hour.} \qquad \text{Formula (15)}$$

Required operating hours (10,000 ÷ 10) ..... 1,000 hr.

Power requirements:

$$\text{Horsepower} = \frac{0.68 \times 10 \times 24}{1{,}000} = 0.1632\text{—say 14 horsepower.} \qquad \text{Formula (16)}$$

Costs:

$RC_{sc} = 0.196 \times 12 \times 24 + 2.333 \times 12 + 0.07 \times \overline{12}^2 = 94.424$—say 96. Formula (17)

$RD_{sc} = (0.061 \times 12 \times 24 + 0.25 \times 12 + 0.007 \times \overline{12}^2)1.15 = 24.81$—say 26. Formula (18)

Take market price at 25 per cent above relative costs.

Cost = $1.25 \times 96 = \$120.00$ ..... initial cost of equipment.

Fixed charges:	(0.085 × 120)	$ 10.20
Labor charge:	(0.0075 × 12 × 1,000)	90.00
Supply expense:	(0.01125 × 0.25 × 1,000)	2.80
Power charge:	(0.03 × 0.25 × 1,000)	7.50
Depreciation:	(1.25 × 26)	32.50
Renewal charge:	(0.20 × 120)	24.00
		$167.00

Net operating cost:
(167 ÷ 10,000) ..... $0.0167 per ton.

Bucket elevators consist of steel or malleable-iron buckets attached to endless chains or belts running about end sprockets or pulleys situated in planes of different elevation. Buckets may be attached in continuous succession (continuous bucket elevators) or, as is more generally the case, at equal spacings (standard bucket elevators). Standard bucket elevators rise in vertical planes or may be slightly inclined, while continuous bucket elevators must be installed at a slight inclination.

Vertical elevators to be self discharging must be of the double-chain type with the return run of the chains snubbed under the head sprockets, or else operated at a speed of from 200 to 960 ft. per minute in order that the discharged material may be thrown over the descending buckets and so avoid undue spill. Continuous bucket elevators and standard elevators without snub pulleys are inclined at an angle with the vertical to avoid the spill and secure correct material discharge.

Buckets for continuous elevators are V-shaped in cross-section and carry their maximum load when the inclination of the elevator is such that the front and rear of the buckets are inclined equally to the vertical. Single-chain and belt elevators usually are operated at about double the speed of continuous bucket elevators so their inclination does not have to be so pronounced. Double-chain elevators when run in a vertical plane have their buckets suspended between the chains, to permit the required snubbing, but inclined elevators of the standard type, when inclined, have their buckets attached to the chains on their backs. Chains should be of the short-pitch variety to avoid pulsations as they pass about the end sprockets and when handling abrasive material likely to raise dust they should be bushed. Supporting sprockets and rollers are used on inclined elevators to prevent the loaded side sinking and interfering with the return run of the elevator.

Elevator boots of cast iron, sheet steel or wood with steel lining and furnished with adjustable take-ups for the boot sprockets or belt pulley are employed with standard bucket elevators. The load is picked up by the buckets as they pass under the sprockets or pulleys in the boots, so the feed to the boot should be at the same rate as that at which the buckets withdraw the material, so as to avoid choking the elevator and flooding the boot. Continuous bucket elevators with their comparative slow speeds permit the load being discharged directly into the buckets.

Bucket-elevator drives, through chains or gears, should be at the head end of the elevators so that the tension side of the elevator should also be the loaded side.

Economic speeds for standard bucket elevators handling various materials are given in Table 11. These speeds are dependent upon the spacing of the buckets being 12, 15, or 18 in., depending upon the size of the buckets. For continuous bucket elevators, the speeds are about one-half those given.

TABLE 11.—ECONOMIC SPEEDS FOR BUCKET ELEVATORS

Standard Type

MATERIAL	SPEED, FEET PER MINUTE	MATERIAL	SPEED, FEET PER MINUTE
Coke	100	Ore (average)	175
Broken stone (coarse)	125	Crushed stone	175
Lump coal	125	Sand and gravel	175
Ashes	150	Fine coal	200
Lime and cement	150		

The carrying (elevating) capacity of bucket elevators depends upon the size of buckets, their spacing, speed of elevator and weight of material handled. When

effectively loaded, the capacity of any type of bucket elevator may be quite accurately ascertained by Formula (19).

$$W = \frac{lw^2Vw'}{9{,}000S} \qquad \text{Formula (19)}$$

Where,

W = Capacity in tons per hour,
l = Length of bucket in inches,
w = Width of bucket in inches,
V = Speed of elevator (buckets) in feet per minute,
w' = Weight of material handled in pounds per cubic foot, and
S = Spacing of buckets in inches.

Bucket-elevator equipment being balanced when not loaded, the power requirements consist in that required to lift the load, overcome the frictional resistance of the apparatus and that required to drag the buckets through the load collected in the elevator boot. The latter consumption of power is not great if the elevator is started before the feed to the boot, as the material is caught in the buckets as they rise in the boot.

$$\text{Horsepower} = \frac{15WH}{10{,}000} \qquad \text{Formula (20)}$$

Where,

W = Load handled in tons per hour, and
H = Height to which load is elevated in feet.

Standardization of bucket-elevator apparatus makes possible the derivation of a general equation for ascertaining the relative cost of such equipment with reasonable accuracy. Elevator buckets are usually either malleable iron or of steel, attached to one or two strands of ordinary detachable-link chain (engineering chain) or chain with malleable-iron links and steel pins (combination chain) or to high-grade rubber belts. Standard bucket elevators have buckets spaced 12, 15 or 18 in. Formula (21) for ascertaining the relative cost of standard elevators with any combination of buckets and chains may also be used for ascertaining the relative cost of continuous bucket elevators by simply substituting an interpolated value for the constant K, based on the depth of the V-bucket.

$$RC_{be} = F\left(\frac{KwlHw'}{1{,}000} + Dwl\right) \qquad \text{Formula (21)}$$

Where,

w = Width of bucket in inches,
l = Length of bucket in inches,
H = Height of elevator in feet,
w' = Weight of material handled in pounds per cubic foot,
K = Constant = 2.70, buckets spaced 12 in.,
= 2.17, buckets spaced 15 in.,
= 1.80, buckets spaced 18 in.,
D = Factor varying inversely with size of bucket 2.1 to 1.1 for standard steel buckets from 5 by 4 in. to 24 by 8 in. inclusive in size, and
F = 1.78, malleable-iron buckets and combination chain,
= 1.57, malleable-iron buckets and engineering chain,
= 1.50, malleable-iron buckets and rubber belt,
= 1.20, steel buckets and combination chain,
= 1.00, steel buckets and engineering chain,
= 0.92, steel buckets and rubber belt.

A similar equation for ascertaining the relative depreciation (annual)—Formula (22)—utilizes other values for constant K and a percentage of the factor D

$$RD_{be} = F\left(\frac{KwlHw'}{10{,}000} + 0.15Dwl\right) \qquad \text{Formula (22)}$$

Where,

K = Constant = 8.0, buckets spaced 12 in.
= 6.8, buckets spaced 15 in.
= 5.6, buckets spaced 18 in.

In charging up the various expenses contracted in operating a bucket elevator, depreciation expenses, which are relatively heavy while the equipment is in use and comparatively light when not in use, need only be charged against operating hours, provided the equipment is in use the greater part of 2,500 hr. during the year.

Fixed charges, consisting of the burden of interest on investment, taxes, insurance, etc., may be taken at 8.5 per cent of the initial cost per year. Labor expense of 5 cts. per inch width of bucket will cover the necessary periodic inspections of equipment as well as the attention required to prevent unwieldy lumps passing to the buckets, etc. On account of the dust which works into the shaft bearings and the chain links and pins, the expense for incidental supplies is high, averaging close to 2 cts. per horsepower consumed.

To illustrate the method of arriving at the total cost of operating a bucket elevator, the following typical example may be taken.

Example.—Required the net operating cost of a 100-ft. double-strand (two chain) elevator handling sand weighing 120 lb. per cubic foot. Elevator to have malleable-iron buckets and combination chains. Power delivered at elevator drive commanding a value of 3 cts. per horsepower. Elevator to be used 1,500 hr. per year. Capacity 25 tons per hour.

Advisable speed of elevator (Table 11) 175 ft. per minute.

$$\text{Capacity, } 25 = \frac{lw^2 \times 175 \times 120}{9{,}000S} \qquad \text{Formula (19)}$$

$$10.7S = lw^2$$

Let $S = 12$, then $lw^2 = 128.4$.

Select 6 by 4-in. bucket.

$$W = \frac{6 \times 4 \times 4 \times 175 \times 120}{9{,}000 \times 12} = 18.66 \text{ tons per hour.}$$

Select 8 by 5-in. buckets, 18-in. spacing.

$$W = \frac{8 \times 5 \times 5 \times 175 \times 120}{9{,}000 \times 18} = 25 \text{ tons per hour, required capacity.}$$

$$\text{Horsepower} = \frac{15 \times 25 \times 100}{10{,}000} = 3.75\text{—say 5 hp.} \qquad \text{Formula (20)}$$

$$RC_{be} = 1.78\left(\frac{1.80 \times 5 \times 8 \times 100 \times 120}{1{,}000} + 1.9 \times 5 \times 8\right) = 1{,}673.2$$

Formula (21)

$$RD_{be} = 1.78\left(\frac{5.6 \times 5 \times 8 \times 100 \times 120}{10{,}000} + 0.15 \times 1.9 \times 5 \times 8\right) = 280.2$$

Formula (22)

Take market price at 25 per cent above relative costs.
Cost = 1.25 × 1,673.2 = $2,091.50, say $2,100.00—initial cost.

Fixed charges:	$(0.085 \times 2{,}100)$	$178.50
Labor charge:	$(0.05 \times 8 \times 1{,}500)$	600.00
Supply expense:	$(0.02 \times 5 \times 1{,}500)$	150.00
Power charge:	$(0.03 \times 5 \times 1{,}500)$	225.00
Depreciation:	$\left(1.25 \times 280.2 \times \frac{1{,}500}{2{,}500}\right)$	210.15
		$1,363.65

Net operating cost:
$(1{,}363.15/1{,}500 \times 25)$ ..... $0.0363 per ton

Bucket carriers are a combination of a bucket elevator and a bucket conveyor in which the path of the buckets includes both elevating and conveying sections. The buckets are pan shaped and attached to two runs of endless chain. They may be attached rigidly to the chains, in which case the buckets are run through troughs on conveying stretches, dragging the load forward much like flight conveyors, or the buckets may be pivoted and suspended between the chains so that they maintain upright positions on both elevating and conveying stretches.

Rigid-bucket carriers discharge their load at points at which the buckets pass about sprockets from elevating to conveying stretches or through gates located in the bottom of the troughs on conveying runs. Pivoted bucket carriers discharge at any point along horizontal stretches of the conveyor by tilting devices which tip the buckets as they pass. Usually there are cams on either side of the buckets which engage a movable tripping cam set where desired.

Buckets are rectangular (depth equals width), pan-shaped, attached to matched endless chains in continuous succession or at regular spacings. The buckets of pivoted carriers are customarily mounted so that the edges of adjacent buckets overlap on horizontal runs, forming a continuous succession of buckets. Chains should be of the short link variety to avoid pulsations and intermittent motion in travel, though long-link chains can be used if compensating sprockets are employed.

Speeds for bucket carriers with rigid buckets may run as high as those employed for flight conveyors, but carriers with pivoted buckets should rarely be run at a speed greater than from 50 to 75 ft. per minute on account of the shock developed when dumping the buckets. Advisable speeds for pivoted bucket carriers when handling various materials are given in Table 12.

Table 12.—Economic Speed of Bucket Carriers for Various Materials—Pivoted Buckets

Material	Speed, Feet Per Minute
Coke	40
Broken stone (coarse)	50
Lump coal—run of mine	50
Ashes	60
Lime and cement	60
Ore (average)	70
Crushed stone	70
Sand and gravel	70
Fine coal	80

The capacity of certain standard sizes of bucket carriers handling material weighing 100 lb. per cubic foot and run at a speed of 50 ft. per minute is given in Table 13,

while Formula (23) is a general equation by which the carrying capacity of any standard size of bucket carrier in tons per hour may be calculated for materials of known weight.

TABLE 13.—CAPACITY OF BUCKET CARRIERS IN TONS PER HOUR

Material weighing 100 lb. per cubic foot. Carrier speed 50 ft. per minute

Bucket length by width, inches	Spacing of buckets			
	18 in.	24 in.	30 in.	36 in.
	Tons per hour			
12 by 12	25.76	19.36	15.41	12.84
16 by 12	34.24	25.68	20.55	17.12
16 by 15	53.60	40.20	32.15	26.80
18 by 15	60.15	45.10	36.10	30 07
20 by 15	.....	52.40	41.90	33 45
24 by 15	.....	60.10	48.00	39.60
20 by 20	.....	88.80	71.00	59.30
24 by 20	.....	106.70	85.40	70.40
30 by 20	.....	.....	106.70	89.10
36 by 20	.....	.....	128.00	106.70
30 by 24	.....	.....	.....	124.60
36 by 24	.....	.....	.....	153.72

$$W = \frac{4.28 w^2 l V w'}{100{,}000\, S} \qquad \text{Formula (23)}$$

Where,

W = Capacity in tons per hour,
w = Width of bucket in inches equals depth of bucket,
l = Length of bucket in inches,
V = Speed of carrier in feet per minute,
S = Spacing of buckets in inches, and
w' = Weight of material handled in pounds per cubic foot.

Bucket carriers, like bucket elevators, are balanced when not loaded, but the calculation of power requirements is complicated by the question of method of conveying the load over horizontal stretches—whether the load is scraped along by rigid buckets or whether the load is carried in buckets running on rollers supported on guide rails. The saving in power on horizontal runs by the pivoted-bucket construction is sacrificed to some extent by the necessity of loading the individual buckets by reciprocating feeders—consuming power—in order to avoid spill of material. Formula (24) is a convenient equation for ascertaining power requirements for either type of carrier, the constants caring for the difference in requirements.

$$\text{Horsepower} = \frac{(AL + BH)W}{100{,}000} + \frac{9W}{100} \qquad \text{Formula (24)}$$

Where,

A = Constant = 28 for carriers with pivoted buckets,
= 103 for carriers with rigid buckets,
B = Constant = 108 for carriers with pivoted buckets,
= 133 for carriers with rigid buckets,
L = Total horizontal span of carrier in feet,
H = Total lift of carrier in feet, and
W = Load handled in tons per hour.

The buckets of carriers representing a considerable item in the cost of bucket carriers, formulas for relative costs and relative depreciation are complicated by the spacing of the buckets, as well as by the size of buckets and the length of the carrier. For the standard spacings of 18, 24, 30 and 36 in., formulas (25) to (28) give results which are quite reliable, and if other spacings are desired interpolation may be resorted to and reasonably accurate results secured.

$$RC_{bc'} = Kwl(H + L)w' + 0.005wlL + 15.25\sqrt{wl} \qquad \text{Formula (25)}$$

Where,

$RC_{bc'}$ Relative cost of bucket carrier with rigid buckets,
K = Constant = 0.00278 for 18-in. spacing
= 0.00208 for 24-in. spacing
= 0.00167 for 30-in. spacing
= 0.00139 for 36-in. spacing,
H = Height to which load is elevated in feet,
L = Horizontal travel of carrier in one direction in feet,
wl = Width by length of bucket in inches, and
w' = Weight of material handled in pounds per cubic foot.

$$RC_{bc''} = Kwl(H + L)w' + (0.05L + 0.75)\sqrt{wl} \qquad \text{Formula (26)}$$

Where,

$RC_{bc''}$ = Relative cost of bucket carrier with pivoted buckets, and
K = Constant = 0.00389 for 18-in. spacing
= 0.00292 for 24-in. spacing
= 0.00233 for 30-in. spacing
= 0.00194 for 36-in. spacing

$$RD_{bc'} = Kwl(H + L)w' + 0.002wlL + 1.525\sqrt{wl} \qquad \text{Formula (27)}$$

Where,

$RD_{bc'}$ = Relative depreciation (annual) of bucket carriers with rigid buckets, and
K = Constant = 0.00066 for 18-in. spacing
= 0.00050 for 24-in. spacing
= 0.00040 for 30-in. spacing
= 0.00033 for 36-in. spacing.

$$RD_{bc''} = Kwl(H + L)w' + (0.005L + 1.6)\sqrt{wl} \qquad \text{Formula (28)}$$

Where,

$RD_{bc''}$ = Relative depreciation (annual) of bucket carriers with pivoted buckets, and

K = Constant = 0.00072 for 18-in. spacing
= 0.00054 for 24-in. spacing
= 0.00043 for 30-in. spacing
= 0.00036 for 36-in. spacing.

On account of the numerous chain joints subject to the abrasive action of dust and fine particles of material, the depreciation of bucket carriers is much more rapid when in operation than when idle, so, provided the hours of operation per year number 60 per cent or more of the available annual working hours—usually taken as 2,500—depreciation per year is customarily taken as chargeable only during actual working hours.

Fixed charges are figured at 8.5 per cent of the initial cost of the equipment per year, while the labor expense averages close to 6 cts. per hour of operation for each inch of bucket width for carriers with rigid buckets and 5 cts. for carriers with pivoted buckets. Incidental expenses for lubricants, etc. average close to 1½ cts. per horsepower consumed.

Net Operating Cost:

Example.—Required the net operating cost of a 500-ft. bucket carrier—lift, 150 ft.; horizontal travel, 350 ft.—handling 120,000 tons of fine coal per year. Carrier to be operated 1,500 hr. per year, with power delivered at carrier drive valued at 3 cts. per horsepower.

Required capacity of carrier (120,000 ÷ 1,500)..... 80 tons per hour.
Advisable speed (Table 12)..... 80 ft. per minute.

Fine coal weighing about 50 lb. per cubic foot and the carrying capacity of a bucket carrier being proportional to its speed, a carrier capable of handling about 100 tons of material weighing 100 lb. per cubic foot when run at a speed of 50 ft. per minute would be required. Select 24 by 20 in. buckets spaced 24 in. Table 13

Check:

$$80 = \frac{4.28w^2l \times 80 \times 50}{100{,}000 \times 24} \qquad \text{Formula (23)}$$

$$w^2l = 11{,}215$$

As $l = 24$, $w^2 = 467.3$, $w = 21.62$ in.

Select 24 by 24 in. buckets, spaced 24 in.

$$\text{Horsepower} = \frac{(28 \times 350 + 108 \times 150)80}{100{,}000} + \frac{9 \times 80}{100} = 28.4\text{—say 30 hp. for pivoted buckets.} \qquad \text{Formula (24)}$$

$$\text{Horsepower} = \frac{(103 \times 350 + 133 \times 150)80}{100{,}000} + \frac{9 \times 80}{100} = 52.0\text{—say 55 hp. for rigid buckets.} \qquad \text{Formula (24)}$$

$$RC_{bc'} = 0.00208 \times 24 \times 24(150 + 350)50 + 0.005 \times 24 \times 24 \times 350 + 15.25\sqrt{24 \times 24} = 31{,}329\text{—say } 31{,}500 \qquad \text{Formula (25)}$$

$$RC_{bc''} = 0.00292 \times 24 \times 24(150 + 350)50 + (0.05 \times 350 + 75)\sqrt{24 \times 24} = 34{,}220\text{—say } 34{,}500 \qquad \text{Formula (26)}$$

$$RD_{bc'} = 0.0005 \times 24 \times 24\ (150 + 350)\ 50 + 0.002 \times 24 \times 24 \times 350 + 1.525\ \sqrt{24 \times 24} = 7639.8\text{—say } 7{,}750 \qquad \text{Formula (27)}$$

$$RD_{bc''} = 0.00054 \times 24 \times 24(150 + 250)50 + (0.005 \times 350 + 1.6)\sqrt{24 \times 24} = 7{,}856.4\text{—say } 7{,}900 \qquad \text{Formula (28)}$$

Take market price at 25 per cent above relative costs.

Cost = $1.25 \times 31{,}500 = \$39{,}375.00$—bucket carrier with rigid buckets.
= $1.25 \times 34{,}500 = \$43{,}125.00$—bucket carrier with pivoted buckets.

		Rigid buckets	Pivoted buckets
Fixed charges:	$(0.085 \times 39{,}375)$.....	\$ 3,346.88	
	$(0.085 \times 43{,}125)$.....	.....	\$ 3,665.63
Labor charge:	$(0.06 \times 24 \times 1{,}500)$.....	2,160.00	
	$(0.05 \times 24 \times 1{,}500)$.....	.....	1,800.00
Supply expense:	$(0.015 \times 55 \times 1{,}500)$.....	1,237.50	
	$(0.015 \times 30 \times 1{,}500)$.....	.....	675.00
Power charge:	$(0.03 \times 55 \times 1{,}500)$.....	2,475.00	
	$(0.03 \times 30 \times 1{,}500)$.....	.....	1,350.00
Depreciation:	$\left(1.25 \times 7{,}750 \times \frac{1500}{2500}\right)$	5,812.50	
	$\left(1.25 \times 7{,}900 \times \frac{1500}{2500}\right)$	.....	5,925.00
		\$15,031.88	\$13,415.63

Net operating cost:

(15,031.88/120,000) $0.125 per ton—carriers with rigid buckets.

(13,415.63/120,000) $0.112 per ton—carriers with pivoted buckets.

Suction conveyors developed originally for handling ashes from the ashpits of boiler furnaces have proved convenient and efficient devices for handling other fine material. The conveying duct is a system of piping from which the air is exhausted by a high-speed fan at the head end of the conveyor. Intakes are situated at loading points and the material handled is swept through the conveying duct and deposited in a receiving hopper located below the exhaust fan by the rush of air through the open intake and through the piping to the exhaust fan. The fine material is carried virtually in suspension and there is little abrasive wear on the walls of the conveying pipe on straight stretches. At elbows, removable wearing plates are necessary to withstand the scouring action of the impinging material, but otherwise the deterioration of the conveying duct is not rapid.

The carrying capacity of a suction conveyor depends upon the diameter of the conveying duct (pipe) and the weight of the material handled. Closed intakes must be tight to avoid leakage of air which would slow down the rush of air and material swept in through feeding intakes and cause the material in suspension to be precipitated. A convenient equation for ascertaining the capacity of suction conveyors in tons per hour is given as Formula 29.

$$W = 0.0385w'd^3 \qquad \text{Formula (29)}$$

Where,

W = Capacity in tons per hour,

d = Diameter of conveyor duct (pipe) in inches, and

w' = Weight of material handled in pounds per cubic foot.

A peculiarity of suction conveyors is that once the necessary degree of vacuum has been created in the system, it requires practically no more power to operate a long conveyor than it does a short one—within reasonable limits. With reasonably tight intakes, the formula for horsepower is given as Formula (30) and it will be noted that the power requirement is quite independent of the load carried, being simply that required to maintain the suction necessary to handle the capacity load.

$$\text{Horsepower} = 0.55d^2 \qquad \text{Formula (30)}$$

The chief items of cost in any suction conveyor system are the powerful exhaust fan required and the tank with its accessories in which the load handled is deposited and these essential appliances are just as costly for a short system as for a long one. The expense entailed for the conveyor duct, intakes, etc., varies, but for all practical purposes the cost of a complete system will be found to vary very nearly directly with its size, diameter of conveyor duct, or conveying capacity. The relative cost of a complete suction conveyor system is then expressed by the equation:

$$RC_{ss} = 143.7d^2 \qquad \text{Formula (31)}$$

Fixed charges amount to 8.5 per cent of initial cost of equipment per year and a flat depreciation rate of 10 per cent per year will cover deterioration, necessary renewals, etc. The labor charge can be overlooked in computing the net operating cost for a suction conveyor system is invariably an adjunct to other operations against which the labor of loading the conveyor, etc., is chargeable, while such little attention as the system may require in the way of inspection, etc., may be charged up to general plant upkeep. Incidental expenses for lubricants, etc., will average about a cent per horsepower consumed.

Example of Net Operating Costs.—Required the net operating cost per ton of a 10-in. suction conveyor handling 7,500 tons of ashes per year, power commanding a value of 3 cts. per horsepower delivered at the exhaust fan drive. Ashes weigh 45 lb. per cubic foot.

Capacity of conveyor:

$$W = 0.385 \times 45 \times 10 \times 10 = 17.325 \qquad \text{Formula (29)}$$

say, 15 tons per hour.

Horsepower requirements:

$$\text{Horsepower} = 0.55 \times 10 \times 10 = 55 \qquad \text{Formula (30)}$$

Costs:

$$RC_{ss} = 143.7 \times 10 \times 10 = 14{,}370 \qquad \text{Formula (31)}$$

Take market price at 25 per cent above relative cost. Initial cost = $1.25 \times 14{,}370 = 17{,}962.5$—say \$18,000.

Fixed charges:	$(0.085 \times 18{,}000)$	\$1,530.00
Power charge:	$(0.03 \times 55 \times 500)$	825.00
Supply expense:	$(0.01 \times 55 \times 500)$	275.00
Depreciation:	$(0.10 \times 18{,}000)$	1,800.00
		\$4,430.00

Net operating cost: (4,430/7,500) ..... \$0.59 per ton

Weights of Certain Solids

Substance	Average sp. gr. water = 1	Average weight lb. per cu. ft.
Asbestos rock	3.2	200
Ashes (packed)	0.72	45
Asphalt	1.1–1.8	68–100
Bone	1.7–2.0	106–125
Brick (hard)	2.0–2.4	125–150
Brick (fire)	2.25–2.4	140–150
Brick soft	1.5–1.7	93–106
Cement, loose	1.3–2.0	81–125
Chalk	2.5	156
Charcoal	.28–.57	17–35
Clay	1.8–2.6	112–162
Coal, anthracite	1.3–1.7	81–106
Coal, bituminous	1.2–1.5	75–94
Coke, solid	1.0–1.7	62–106
Coke, loose piled	.....	20–30
Corundum	3.9–4.0	243–249
Dolomite	2.8	177
Earth, dry	.....	76–95
Earth, moist	.....	78–98
Feldspar	2.55–2.75	159–172
Flint	2.6	162
Gas carbon	1.88	117
Granite	2.56–2.88	159–189
Graphite	2.26	141
Gravel	1.6–1.9	100–120

WEIGHTS OF CERTAIN SOLIDS.—*Continued*

Substance	Average sp. gr. water = 1	Average weight lb. per cu. ft.
Greenstone (trap)	2.7–3.2	170–200
Gypsum	2.32	145
Lignite	1.3	80
Lime, quick	1.5	94
Lime, slaked	1.3–1.4	81–87
Limestone	2.7	168
Marble	2.5–2.8	160–180
Marl	1.6–2.25	100–140
Mica	2.8	175
Mud	1.67–1.92	104–120
Peat	.84	52
Pumice	.92	57
Quartz, broken	.....	94
Salt, massive	2.1	128
Sand	.....	100–130
Sandstone, massive	2.14–2.36	133–147
Slag, crushed	1.2	75
Slate	2.6–3.3	162–205
Sulphur	2.05	128
Traprock	3.0	187

NOTE.—Unless specified otherwise, the figures are for the materials in the mass. When moved crushed or loose, there will usually be 25–35% of voids.

SECTION III

THE TRANSPORTATION OF LIQUIDS

By F. F. Nickel[1]

Introduction.—In industrial establishments the means of elevating and transporting liquids may be roughly classified into piston pumps and pistonless pumps, the latter including pulsometers, jet pumps, air lifts, eggs or montejus, centrifugal pumps and rotary pumps.

PISTON PUMPS

Single direct-acting pumps have one steam (or air) cylinder and one pump body, both double-acting. The essential part of the valve motion is a pilot valve controlling a valve piston which mechanically operates the main steam valve. The main ports terminate a short distance from the end of the cylinder and a small port relieves the compression and admits steam behind the piston for starting, see Fig. 1. Advantages over duplex pumps are: greater simplicity, as only two stuffing boxes and one pump piston have to be cared for; also somewhat higher economy because the piston must make a full stroke before it throws the pilot valve. This is of especial advantage in wet vacuum pumps, as it

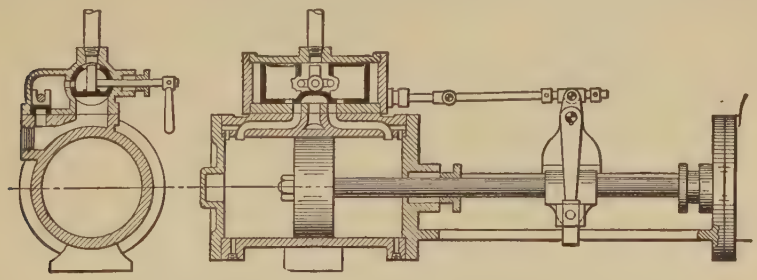


Fig. 1.—Laidlaw-Dunn-Gordon horizontal valve motion.

reduces the clearance. A disadvantage is the intermittent action of the plunger, resulting in shocks in the pipes, which must be taken up by large air chambers. These shocks are especially violent at higher speeds and when the liquid carries much entrained gas. Plunger and ring pumps should be used for clean water only, as grit and dirt rapidly destroy the fit and cause leakage.

Simple cylinder direct-acting duplex pumps are widely used for all purposes. Figure 2 shows a standard design. It consists of two steam cylinders and two water ends, arranged side by side. Each steam piston is directly connected to its

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water piston. The steam piston of one side operates mechanically the valve of the opposite side. A certain lost motion, or valve rod clearance, M, is provided in the transmitting mechanism. This retards the motion of the valve and allows the steam piston to finish its stroke. The steam cylinder is equipped with double ports, the outer one being the steam port and the inner one the exhaust port. The piston in approaching the head closes its own exhaust port, entraps the steam and by compressing it is stopped by this steam cushion. Larger pumps are provided with valves for regulating the cushion.

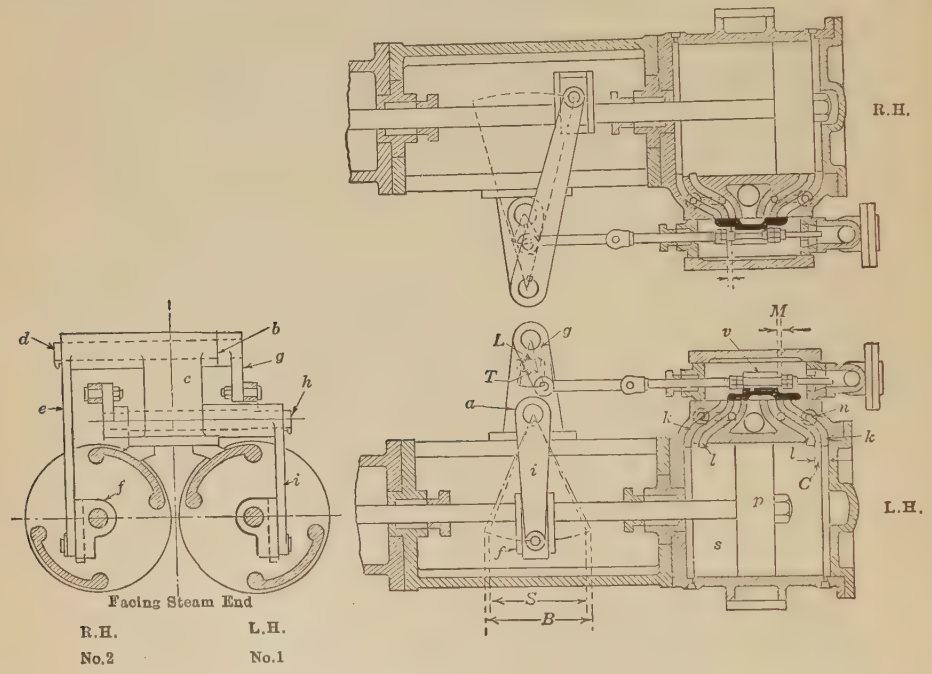


FIG. 2.—Simple cylinder direct-acting duplex pump.

Duplex pumps give an almost uniform flow of water and require only small air chambers. The speed is limited and those given in Table I should not be exceeded. The economy is low, because no cut-off is possible so that the steam is exhausted at full pressure and its expansive energy is not utilized.

Compound duplex pumps are direct-acting pumps with a high- and a low-pressure steam cylinder arranged tandem on each side and connected directly to the water end. The high-pressure steam cylinder takes steam full stroke, and the steam then exhausts through the intermediate space into the low-pressure cylinder. In this way part of the expansive force of the steam is utilized. Figure 3 shows a longitudinal section through a compound steam end and an end view with a cross-section in the high-pressure cylinder. Packing between the cylinders is here provided by a close-fitting floating sleeve. Stuffing boxes may be used for the same purpose. The intermediate spaces, consisting of the high-pressure exhaust pipes, the low-pressure steam chests and the ports, are connected by the cross exhaust pipe b, with a valve c inserted for adjustment.

TABLE 1.—SPEEDS OF PUMPS

Stroke	Direct-acting pumps							Geared power pumps	Crank and flywheel pumps also belt or direct-driven power pumps
	Piston pumps, plunger and ring pumps, outside packed pumps up to 300 lb. pressure				Pressure-pumps, wet vacuum pumps	Boiler-feed p. and pumps for handling thick liquids	Direct-acting Water motor driven pumps		
	Simple cyl. and Compound		Triple expansion						
Inches	Cycles per minute	Feet per minute	Cycles per minute	Feet per minute	Feet per minute	Feet per minute	Feet per minute	Feet per minute	Feet per minute
3	80	40	..	...	32	20	13	32	
4	75	50	..	...	40	25	17	40	
5	72	60	..	...	48	30	20	48	
6	65	65	..	...	52	33	22	52	
8	56.2	75	..	...	60	38	25	60	
10	48	80	54	90	64	40	27	64	
12	45	90	50	100	72	45	30	72	180
15	40	100	44	110	80	50	33	80	200
18	36.7	110	40	120	88	55	37	88	220
24	30	120	32.5	130	96	60	40	96	240
30	25	125	28	140	100	63	42	100	250
36	22.5	135	25	150	108	68	45	108	270
48	18	145	20	160	116	73	48	116	290
60	15	150	17	170	120	75	50	120	300

For definition of cycle, see p. 135.

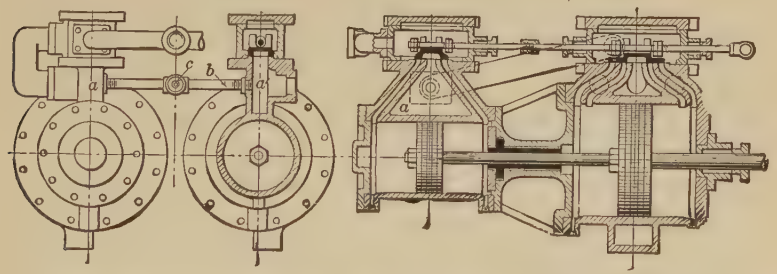


FIG. 3.—Sections of compound steam end showing cross exhaust and floating sleeve.

The steam in expanding from the high-pressure cylinder into the low-pressure cylinder is reduced in pressure, resulting in a declining propulsion curve and short stroking. By opening the cross exhaust valve *c* the size of the intermediate space is doubled and, therefore, the high-pressure-exhaust steam expands to a lower pressure when flowing into the intermediate space. The low-pressure initial pressure is thus reduced while the terminal pressure remains the same. This results in a more uniform back pressure in the intermediate space, and a more nearly straight propulsion curve, which is the aim of the designer of a direct-acting pump, since the load line of a pump is straight. In this way the stroke is lengthened but at the expense of power.

Single pumps are also tandem-compounded, but either the ratio of cylinders is less than for a duplex pump, due to the lack of the cross exhaust or a large receiver is inserted which serves a similar purpose as the cross exhaust.

Triple expansion direct-acting steam pumps have three cylinders on each side, arranged tandem, *viz.*: one high-pressure, one intermediate-pressure, one low-

pressure cylinder. There are two cross exhaust valves, one on each exhaust pipe, affording very close adjustment. The high-pressure cylinder is equipped with cut-off valves and one of the others is provided with cushion ports and valves. The economy is very high, a large part of the expansive force being utilized. Disadvantages are the great number of cylinders, stuffing boxes, and valves, the large floor space required and also the room needed for taking out the rods. Advantages over crank and flywheel engines are the simplicity of the valve motion, the light foundation required, the low cost of repairs and attendance and the uniform discharge. Figure 4 shows a longitudinal section through the steam end, fitted with rotary valves. Adjustment for speed is usually made by hand

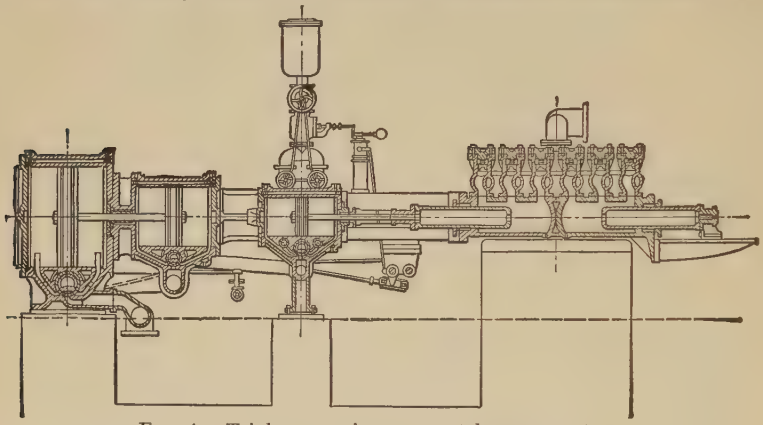


Fig. 4.—Triple-expansion pump, telescope pattern.

throttle valve, but a Mason speed governor controlling a balanced valve in the steam pipe may be attached. To prevent racing in case of accident to the discharge main an automatic stop valve, normally held open by the water pressure, is recommended. This type of pump should be used when the resulting saving in fuel justifies the increased initial cost.

High-duty pumps are equipped with steam cylinders in which the steam is used expansively. As the load is constant throughout the stroke and the steam piston force diminishes during the period of expansion, some compensating device is necessary to equalize the forces. This may be a fly wheel, an auxiliary piston, a weight or a compensating gear.

The compensating device used in the Worthington high-duty direct-acting pump consists of two oscillating hydraulic cylinders, arranged so as to take up the excess of energy developed during the first half of the stroke and return it during the second half, where there is normally a deficiency. To reduce the oscillating masses, high water pressure (600 lb.) is employed. This is furnished by an intensifier which has a large air cylinder supplied from the air chamber and a small hydraulic ram.

The low-duty triple-expansion pump has taken the place of the high-duty compound pump and only triple-expansion high-duty pumps are now built. They give the same duty as crank and flywheel triple-expansion pumps, have greater flexibility, require lighter foundations and the valve motion is simple. Adjustment for speed is made by hand throttle valve which is satisfactory because the load is constant.

Power pumps are pumps having plungers operated mechanically from a rotating driving mechanism. There may be from one to five plungers, arranged

side by side and connected in parallel, driven by a crankshaft with cranks spaced equally around a circle. They may run at high speeds if direct-connected to a motor or belt driven, but if geared, the speed is generally kept low (see table I). The plungers may be single-acting or double-acting. Triplex power pumps having 3 single-acting plungers give an almost uniform flow of water.

Crank and flywheel pumps are power pumps driven directly from the shaft of a steam engine. The horizontal direct-connected type consists of a regular steam engine with a water end attached to the rear steam-cylinder head. The opposed type has the crankshaft placed between steam end and water end. Any combination of the well-known types of steam or water ends may be used. The piston speed seldom exceeds 300 ft. per minute (see Table I). Vertical crank and flywheel pumps may be cross-compound with two double-acting plungers or triple-expansion with three single-acting plungers. The latter gives much higher economy also a nearly uniform flow of water, and the pump end is simple and accessible.

Express pumps, or high-speed power pumps are built for direct connection to electric motors. A Knowles express pump having two double-acting end-packed plungers, 3½-in. diameter by 5½-in. stroke. has a capacity of 250 gal. per minute against a head of 1,000 ft. at a speed of 300 r.p.m. Pumps of this type are built in capacities of from 200 to 4,000 gal. per minute against heads from 100 to 2,000 ft. Two duplex express pumps, having two double-acting end-packed plungers, $6\frac{1}{16}$-in. diameter by 15-in. stroke, were installed for unwatering the Comstock Lode, Virginia City, Nev. The capacity was 1,600 gal. per minute against 1,550 ft. head, speed 195 r.p.m. or a piston speed of 488 ft. per minute. The valves were of the automatic poppet form, 13 in each set, the valve area was 103.35 sq. in. or 294 per cent of the plunger area, giving a water velocity of 165 ft. per minute through the valve seats. Each pump weighed 300,000 lb. and cost $80,000. The cost including the motors was 13.3 cts. per pound, the cost per gallon of capacity $25, the total installation was estimated at $125,000 or $40 per minute-gallon.[1]

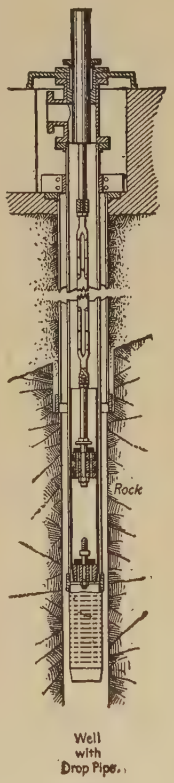


FIG. 5.

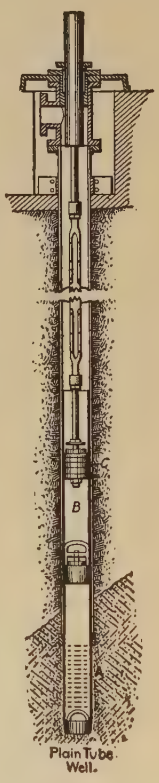


FIG. 6.

Deep-well pumps.

Deep-well pumps of the bucket type are designed for use on non-flowing wells where the water does not stand within suction distance. Two methods are employed: In the **plain tube well** (Fig. 6) the well pipe with open ends is sunk to the proper depth where a sufficient water-bearing stratum is penetrated. The strainer *A* is then lowered to bottom of the well and the well pipe drawn back far enough to expose the slotted portion. The working barrel *B* is then lowered into the gum packer on top of strainer and tapped firmly into place. The top of the well pipe is provided with a tee for discharge connection. The

[1] At pre-war-prices. EDITOR.

bucket C is driven by means of wooden rods of convenient length (18 ft.), coupled together. The top of the well pipe is provided with a stuffing box through which a brass plunger works. If the pump discharges at the surface this plunger is made one-half the area of the bucket, resulting in a uniform discharge. If the pump discharges into an elevated tank the top plunger is made larger so as to throw the effect of the additional head upon the plunger and thereby assist in the up stroke. The **well with drop pipe** (Fig. 5) differs from the plain tube well in that the brass working barrel is attached to a pipe and is lowered into the well pipe. This is a better arrangement in case of a small supply, as it allows a head of water to collect above the bucket during the down stroke. It is used in wells that are either wholly or partly drilled into rock and in wells where the casing does not extend low enough to reach the water supply; also in old wells with defective or bent casing.

CENTRIFUGAL PUMPS

A centrifugal pump consists of a casing in which revolves an impeller, that is a wheel carrying a number of suitably shaped vanes. The centrifugal force sets up a static head in the casing and a certain velocity is imparted to the liquid. With the discharge valve closed only the static head is noticeable and can be measured. This is called the shut-off head and is $H_s = \frac{u_a^2}{2g}$ nearly, where u_a = the peripheral velocity of the impeller in feet per second and $g = 32.2$ ft. per second. When the discharge valve is opened other forces come into action and it is possible to obtain a head higher than that due to the centrifugal force alone. This higher head is chiefly obtained by the conversion of the velocity into static head.

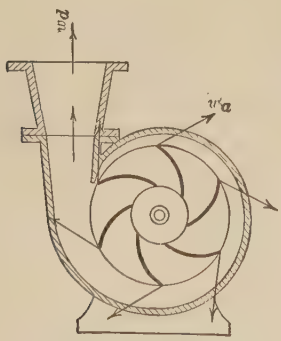


FIG. 7.—Volute pump.

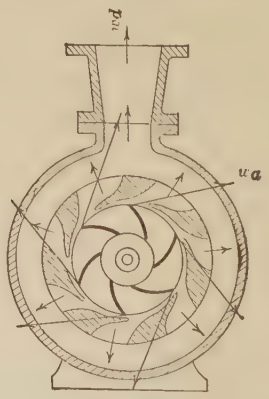


FIG. 8.—Turbine pump.

This conversion can be accomplished in two ways, resulting in the two types:

(*a*) *Volute Pump.*—The impeller is surrounded by a spiral casing (the volute) of gradually increasing cross-section, Fig. 7.

(*b*) *Turbine Pump.*—The impeller is surrounded by a guide ring, provided with a number of suitably shaped channels which receive the liquid in the direction in which it leaves the impeller, guide it to the outside and at the same time reduce its velocity to such an extent that it will readily turn in any direction and flow towards the point of the casing where the discharge may be located, Fig. 8. The volute

pump is recognized by its spiral casing and tangential discharge nozzle, the turbine pump by its concentric casing and radial discharge nozzle.

Velocity Diagram.—The water at the point of exit from the impeller has two velocities, *viz.:* The relative velocity v_a, tangential to the tip of the vane, and the peripheral velocity u_a, tangential to the periphery of the impeller. These two velocities combine into the absolute velocity, which is indicated by the line marked w_a in Fig. 9. The liquid leaves the impeller and enters the volute casing in the direction of this line w_a. Here it encounters liquid flowing around the impeller in a more tangential direction and the results are large hydraulic losses, due to shocks, eddies and cross currents. These losses are more or less completely avoided in the turbine pump, hence its greater efficiency and the possibility of using higher velocities. The practical limiting velocities and heads in a volute pump with one impeller are $u_a = 120$ ft. per second with a corresponding head of about 220 ft., while with a well-designed turbine pump u_a may be = 140 ft. per second with a corresponding head of about 300 ft.

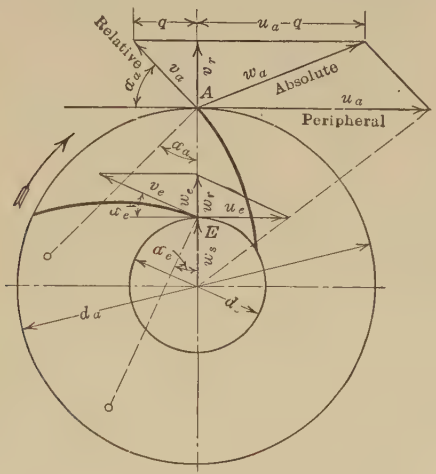


Fig. 9.—Centrifugal pump velocity diagram.

Higher velocities than the above are not practicable as the losses increase a the square of the velocity, and besides the high velocity may result in pitting and excessive wear of the impeller.

The top of the vanes at the point of exit may be curved backward or forward, or may be radial. The first is used almost exclusively at the present date. Forward curved vanes are only used when it is desired to impart to the water a high velocity rather than pressure.

In a similar way the "entrance diagram" can be shown which gives the relation of the various velocities to each other at the point of entrance to the impeller. The entrance diagram shown in Fig. 9 is the ideal entrance diagram which may differ considerably from the actual entrance diagram. In the former the water is assumed to enter the impeller in a radial direction, w_s representing the velocity of approach which is maintained as w_e also in a radial direction. The radial component w_r is then also = w_e.

The water in flowing through the suction opening, however, will partake of the rotary motion of the impeller and will approach the point E, Fig. 10, at some unknown angle. The correct entrance diagram will then appear like that shown in the latter figure. As the angle is not known the angle of the vane is selected to suit the shape of the vane and by experience. In any case the vane must be curved backward at point E.

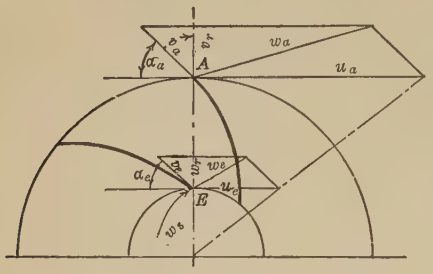


Fig. 10.—Centrifugal-pump velocity diagram with actual entrance diagram

The development of the fundamental equation is based on the ideal entrance diagram.

Multi-stage Centrifugal Pumps. For higher heads two or more impellers may be arranged in series, the first impeller taking suction from the well and discharging the water into the suction of the second impeller, etc. The impellers are mounted on the same shaft and built into one casing, designed to form suitable channels.

In order to reduce the leakage of water from the discharge to the suction side of the impeller wearing rings are used. Preferably they are made separate from the impeller so that they may be renewed if worn. They also serve to balance the outer part of the impeller, both sides of which will be subject to the same pressure. The

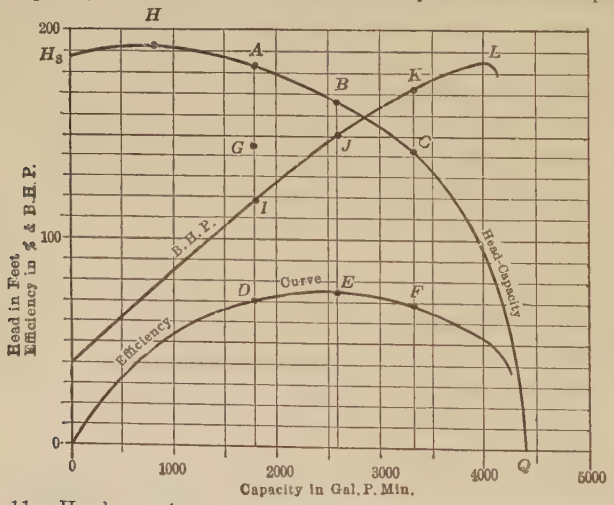


Fig. 11.—Head-capacity, power and efficiency curves for centrifugal pumps.

inner part of the impeller is balanced by coring holes through the web near the hub so that the leakage may return to the suction and prevent the pressure on the back from building up. This equalizing is not perfect and, therefore, a thrust bearing must be provided to take care of the unbalanced axial thrust.

By the use of double-suction impellers almost perfect balancing is accomplished. The design of a single-stage double-suction impeller pump is simple, see Fig. 12. Here the suction branches out so as to embrace the casting, and the water enters from both ends and is discharged in the center. Multi-stage pumps with double-

suction impellers are more complicated in design, and the water must be brought to the opposite side of the impeller through holes cored in the guide ring between the channels. As this may introduce a slight difference in pressure and cause an unbalanced axial thrust a small thrust bearing is provided. In the single-stage pump two collars are merely placed next to the bearing to keep the impeller in place.

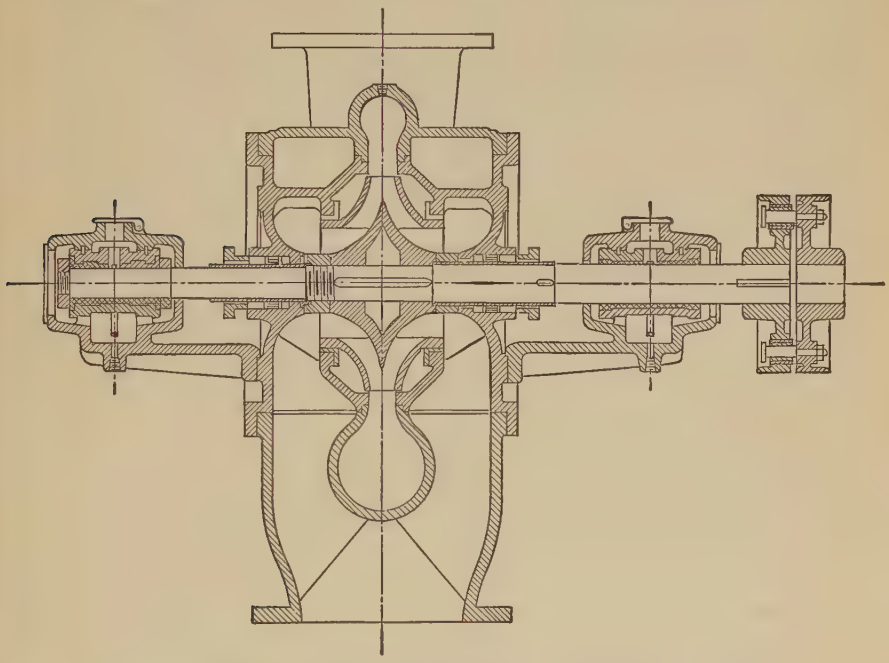


FIG. 12.—Single-stage centrifugal pump with double-suction impellers.

Hydraulic balancing devices secure perfect balancing, but they are only recommended for clean water. The single-suction impellers have wearing rings only on the suction side to prevent short circuit losses. The leakage from the last impeller enters the space between two discs one of which is fastened to the casing and the other to the shaft. Wearing rings on the periphery of these two discs prevent the leakage from escaping until a pressure is set up between the two discs to counteract the end thrust. This is also used on vertical pumps to support the part of the weight of the rotor that is not counterbalanced by the end thrust.

Theory of the Centrifugal Pump.—The *fundamental equation* is

$$u_a = v_r \div 2 \tan \alpha_a + \sqrt{gH \div E_h + (v_r \div 2 \tan \alpha_a)^2}$$

where, referring to Figs. 9–10, u_a = peripheral velocity, feet per second, v_r = radial component of the absolute velocity w_a, α_a = vane angle, degrees, g = 32.2 ft. per second, H = actual head pumped against, feet, E_h = hydraulic (or manometric) efficiency = actual head ÷ ideal head, or $Hg/(u_a - q)u_a$, $u_a - q$ being the tangential component of the absolute velocity w_a (see Fig. 9). Cf. "Hydraulic Efficiency" for values. In testing a centrifugal pump four sets of readings are taken: Capacity, Q; Head, H; Speed, N; and Power, P. The observed test points are then plotted, and thus certain curves are obtained called the *characteristics* of the pump. Of these the HQ or Head-capacity curve is the

most important. From the HQ points the water horsepower can be calculated and by comparing this with the observed brake horsepower the pump efficiency can be computed and the EQ or efficiency-capacity curve added to the diagram.

These three curves are shown in Fig. 11. The test is started with the discharge valve wide open, which produces the maximum capacity against practically no head. This condition is indicated by point Q, corresponding, in the case illustrated, to a capacity of 4,400 gal. per minute. The discharge valve is now gradually closed, whereupon the capacity diminishes and the head rises until it reaches a maximum value at H = 193 ft. From this point on it decreases until, when the discharge valve is closed the shut-off head, H_s = 189 ft., is reached at the point of no capacity. Under both conditions given by the two extremes, *viz.:* Head = 0 and Capacity = 0, the Efficiency = 0. Between these extremes the efficiency curve rises to a maximum value, in this case, E = 74 per cent. Generally this maximum efficiency is only obtained for one set of conditions, in this case, when H = 168 ft. and Q = 2600 gal. per minute. For any other condition the efficiency will be less and for this reason an efficiency of probably 70 per cent would be guaranteed. This guaranteed efficiency would then hold good for a range of conditions, indicated by the line between points A and C. By partly closing the discharge valve the normal head in the column pipe may be maintained for quite a range in capacity. Thus, if the normal head was 145 ft. a capacity of 3,300 gal. per minute (point C) could be obtained, but by throttling the discharge it would also be possible to run at point G, giving a head of 182 ft. in the pump while maintaining the head of 145 ft. in the column without dropping below the efficiency of 70 per cent. Under these conditions the capacity would then be 1,800 gal. per minute so that by merely throttling the discharge any capacity between 1,800 and 3,300 gal. per minute against a head of 145 ft. with an efficiency of 70 per cent can be had.

The three factors: speed, capacity and head are intimately related to each other, and the relation is given by the fundamental equation. The speed is represented by u_a, the capacity is proportional to v_r, and H is the head pumped against. In the following a few rules will be given for analyzing the performance of a pump: In a given impeller the vane angle a_a is known, and the factors u_a, v_r, H and E_h can be computed from the test results by using the following formulas: $u_a = d_a\pi n \div 12 \times 60 = d_a n \div 229.2$; $v_r = Q \div \text{area} = Q \div 12(d_a\pi - zs_a \div \sin a_a)b_a$; $E_h = Hg/(u_a - q)u_a$ where d_a = external diameter of impeller, inches, n = revolutions per minute, Q = capacity in cubic inches per second, z = number of vanes, s_a = thickness of vanes, inches at exit, g = 32.2 ft. per second, q = the horizontal component of the velocity v_a in feet per second = $v_r \div \tan a_a$, and b_a = the width of impeller at exit, inches.

Example.—The pump to be tested has an impeller of the following dimensions: d_a = 21.375 in., b_a = 1 1/16 in., z = 6, s_a = 0.25 in., a_a = 18.5° and is running at the speed of 1,100 r.p.m. The test results are plotted in Fig. 11. The maximum head obtained was H_m = 193 ft., the maximum efficiency E = 74 per cent, and the head at maximum efficiency was H = 168 ft. The capacity at the point of maximum efficiency was Q = 2,600 gal. per minute = 10,010 cu. in. per second. The maximum capacity was Q_m = 4,400 gal. per minute. By substituting these values based on actual test in the above formulas, we can compute the various factors composing the fundamental equation, *viz.:*

$$u_a = 21.375 \times 1{,}100 \div 229.2 = 102.6 \text{ ft. per second}$$
$$v_r = 10{,}010 \div 12(21.375\,\pi - 6 \times 0.25 \div \sin 18.5°)1.56 = 8.56 \text{ feet per second.}$$
$$q = 8.56 \div \tan 18.5° = 25.6 \text{ feet per second}$$
$$u_a - q = 102.6 - 25.6 = 77 \text{ ft. per second}$$
$$E_h = 32.2 \times 168 \div 102.6 \times 77 = 68.5 \text{ per cent.}$$

To test the fundamental equation we will substitute therein the above computed values: and find the speed:

$u_a = 8.56 \div 2 \tan 18.5° + \sqrt{32.2 \times 168 \div 0.685 + (8.56 \div 2 \tan 18.5°)^2} = 102.6$ ft. per second.

The calculated shut-off head would be $H_s = \frac{u_a^2}{2g} = \frac{102.6^2}{64.4} = 164$ ft. while the actual shut-off head was 189 ft. showing that the above rule is only approximate.

The hydraulic efficiency is the ratio of the useful power delivered in the water ($= WH$) to the power delivered to the water by the impeller. The latter is equal to the torque multiplied by the angular velocity w. The torque is the mass W/g multiplied by the radius r_a and the tangential component $u_a - q$ of the absolute velocity w_a, see Fig. 11. Hence,

$$E_h = WH \div (W/g) r_a (u_a - q) w = Hg \div (u_a - g) u_a.$$

Approximate rules: For short vanes, or open impellers: $E_h = 30$ per cent, for Volute Pumps $E_h = 50$ to 60 per cent, for Turbine Pumps $E_h = 57$ to 69 per cent.

TABLE 2.—APPROXIMATE RELATION BETWEEN v_r/u_a AND a_a

	Constant head			Variable head		
	a_a	18°	20°	25°	30°	35°
Volute pump	$\frac{v_r}{u_a}$	....	0.125	0.15	0.175	0.2
Turbine pump	$\frac{v_r}{u_a}$	0.08	0.085	0.09	0.095	0.1

In example $v_r = 8.56$ ft. per second, and $u_a = 102.6$ ft. per second. This gives $v_r \div u_a = 0.08$, for which an angle of 18.5° was selected.

For a head of 200 ft. v_r may be taken as high as 15 ft. per second.

Peripheral Velocity and Rotative Speed

$$u_a = \frac{\pi d_a n}{12 \times 60} = \frac{d_a n}{229.2}; \qquad n = \frac{229.2 u_a}{d_a}$$

Where n = revolutions per minute, d_a, external diameter, inches
u_a = peripheral velocity in feet per second.

The efficiency of a centrifugal pump falls off quite rapidly at speeds and capacities greater or less than those for which the pump is designed. Maximum efficiencies as high as 85 per cent have been attained under low heads (50 ft.), decreasing to about 75 per cent for high speed and high head (100 to 150 ft.) in single stage pumps and to from 55 to 70 per cent in two- and three-stage pumps for high lifts.

Overloading the Motor.—The usual characteristics show a rising power line, and it is important that the motor be selected that can take care of the maximum load. Thus in Fig. 11, although the normal conditions may call for only 149 hp. (point J), the maximum is 185 hp. (point L).

The advantages of centrifugal pumps over reciprocating pumps lie in their lower initial cost, smaller floor space requirements, less need for attendance, quiet operation, absence of excessive stresses in the pipe lines owing to the uniform discharge and adaptability for being driven by motors of high rotative speed. Silt or grit interferes but little with their working. The disadvantages are a certain lack of flexibility

in regard to capacity, head and speed and the reduced efficiency under conditions other than those for which the pump was designed.

PISTONLESS PUMPS

Pulsometers

Pulsometers (Fig. 13) are pistonless steam-operated displacement pumps with two chambers, *AA*. The steam enters these chambers alternately and is controlled by a ball or flap valve *C*. It enters one chamber and forces the water contained therein out through a check valve *F* to the discharge pipe *H* until the steam enters the discharge pipe where it is condensed and produces a suction effect which throws the valve *C* and draws up the water through the suction valves. *J* is an air chamber. Notwithstanding their low efficiency they are used extensively in mines, at railroad watering stations, for draining building pits, etc. Suction lift, up to 26 ft., max., preferably 7 to 14 ft.; discharge head, up to 150 ft. The steam pressure must be about 50 per cent higher than the total water pressure. Low heads give a lower efficiency than high heads. The discharge water is heated 3.5°F. for a head of 30 ft. For every additional 30 ft., add 1.5°. For water-temperatures in excess of 120°F., no suction lift is obtainable. The steam consumption by volume is from 2 to 3 times the water displacement. Wood finds **duties** varying from 10.5 to 13.4 million ft.-lb., the higher figure corresponding to the higher head. This corresponds to a steam consumption of 190 to 150 lb. per horsepower per hour. A duty of 21,345,000 ft.-lb. has been found for total lifts of 102.6 ft.

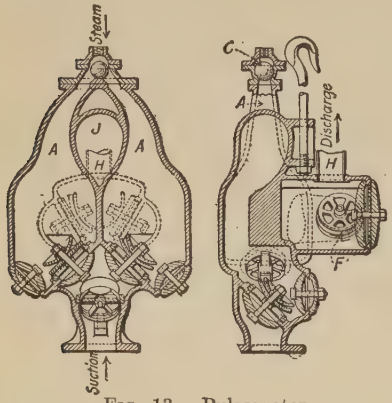


FIG. 13.—Pulsometer.

Compressed air pumping may be accomplished by displacement apparatus (eggs, montejus, etc). The air pressure is applied directly on the liquid in the tank. Two displacement tanks are generally used, alternately filling and discharging. The valve distributing the compressed air may be operated by hand, floats, by the drop in air pressure upon discharge of liquid contents, the air following the liquid through the line, or some suitable timed mechanism. If sufficient submergence can be had the tanks are submerged and the liquid will

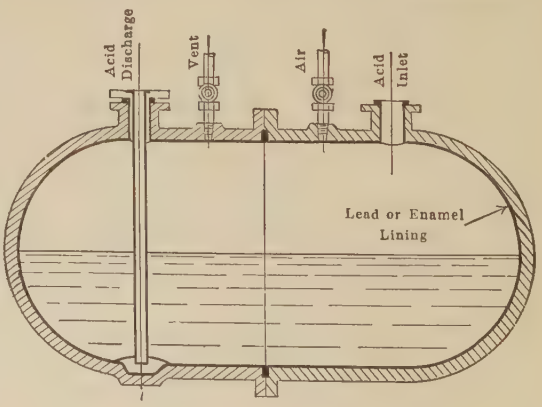


FIG. 14.—Acid egg.

then flow in by gravity, as in the Wheeler system. If not the empty tank may be connected to the suction side of the air compressor and a partial vacuum produced, as in the Harris system. The acid egg is of particular advantage in chemical establishments for the cheap intermittent handling of highly corrosive fluids.[1]

The Starrett air lift pump described in Eng. and Min. Journ., Mar. 30, 1907, utilizes two displacement chambers from which the air alternately displaces the water by pressure, after which the same air is allowed to escape up the stand pipe and, in the manner of the air lift, assists in raising the water and increasing the effective head. A test of such a pump by H. C. Behr (*Cal. Min. Bur., Bull.* 9) gave an efficiency of 50 per cent at the compressor, and 35 per cent net.

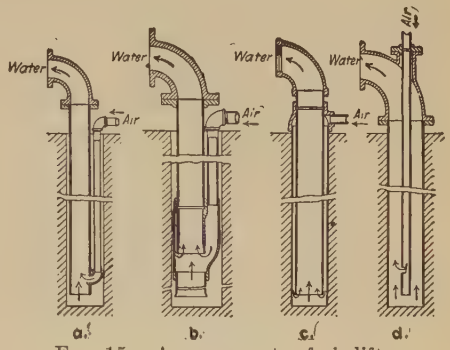


FIG. 15.—Arrangements of air lifts.

The air lift consists of a drop pipe placed in a well with its lower end submerged, so that it is under the free pressure of the water more or less filling the well. An air pipe delivers air at the bottom of the drop pipe and forms a mixture of air and water which is lighter than the column of solid water in the well; consequently the mixture rises above the level of the surrounding water. The displacement of the air compressor required per gallon of water pumped is $D = kl/585 \log [1 + (s/34)]$ cu. ft., where l = lift in feet, measured from the lower water level, when pumping, to the point of discharge, s = submergence below the lower water level, feet when pumping, k = a coefficient taking care of all the losses, approximately $k = 3 + (l/700)$. The operating pressure = s ft. water col., is not always known in advance, as the water will stand at a lower level when pumping than when not pumping, and it is the former that determines the point from which the submergence is measured. The central air pipe system, although obstructing the passage to some extent, has the advantage over the other systems in that the submergence can be changed con-

TABLE 3.—ECONOMICAL CAPACITIES AND DIMENSIONS OF AIR LIFTS WITH CENTRAL AIR PIPE

Diameter of			Gallons per minute based on a velocity of 4 ft. per second
Well, inches	Drop pipe, inches	Air pipe, inches	
4	3	1	70
5	3½	1¼	95
6	4	1¼	125
6	4½	1½	158
7	5	2	195
8	6	2½	280
9	7	2½	380
10	8	3	500
12	10	4	780

[1] This apparatus is a container of oval shape similar to Fig. 14. Such an egg-shaped vessel is free from pockets and is readily drained.

veniently and thus the point of highest economy determined. The starting pressure $= S_a$ ft., where S_a = submergence when not pumping, is generally higher than the operating pressure. In Fig. 15, a = Pohle or side inlet, b = annular space inlet, c = Saunders system, d = central air-pipe system.

The efficiency of the air lift is very low, but its many advantages, *viz.* simplicity, absence of working parts in the well, ability to handle gritty water or oil, and principally the fact that the full area of the well may be rendered available tend to counteract its low duty. Generally it is desirable to work a well to the limit of its capacity, regardless of expense.

TABLE 4.—AIR-LIFT DATA[1]

Lift, feet, l	Water horsepower, W	Displacement of air compressor, cubic feet per minute, D	Operating air pressure, pounds per square inch	Air, horsepower, adiabatic, A	Indicated steam horsepower, I	Over-all efficiency, per cent, $100\frac{W}{I}$
20	0.00505	0.38	13	0.017	0.0213	23.7
40	0.0101	0.475	26	0.036	0.045	22.5
80	0.0202	0.68	52	0.0825	0.103	19.6
120	0.0303	0.82	78	0.129	0.162	18.8
160	0.0404	0.97	104	0.178	0.223	18.2
200	0.0505	1.13	130	0.235	0.294	17.2
250	0.063	1.44	162	0.335	0.418	15.0

The best efficiency is obtained for a capacity of from 10 to 15 gal. per minute per square inch of drop pipe (allowing for an obstruction of 20 per cent by the central air pipe, this will be equivalent to a water velocity of from 3.2 to 4.8 ft. per second.

Another method of air pumping is to admit compressed air to the steam end of a steam pump. This can be done without difficulty in a simple cylinder direct-acting pump, but where the air is used expansively, the moisture in the air is apt to freeze at the exhaust nozzle and choke the exit. A remedy is to heat the air before admission, and this also effects a considerable increase in efficiency which at best is quite low.

The hydraulic ram is used for lifting small quantities of water against high heads, where an abundance of drive water is available. The usual capacities are from 1 to 30 gal. per minute. The drive water flows through a pipe and out through a clack valve. As soon as a certain velocity is attained the valve closes and the water continues to flow by inertia and rises to a higher level in the discharge pipe. The operation continues automatically. In practice it is impossible to lift water by a ram to a height over 20 times the head H. The drive pipe should be unobstructed and have a length in feet $= (H + h + 2)2h \div H$, and not less than $= 5H$. An average velocity of 0.8 ft. per second should be secured in the drive pipe. The volume of the air chamber should be approximately equal to that of the discharge pipe. There are two efficiency formulas used for rams, one by Rankine, giving the commercial efficiency $E = qh \div QH$,

[1] Based on a submergence of $1.5l$, this being considered the most desirable submergence. Figures in table are for a capacity of 1 gal. per min.

in which q is the quantity of water raised h ft. above the source, and Q is the quantity of water wasted in falling H ft. to do the work. The efficiency under the Rankine formula is limited by the ratio of lift to fall, approximately as follows:

h/H	4	6	8	12	16	20	24	26
E	0.72	0.61	0.52	0.37	0.25	0.14	0.04	0

The d'Aubisson formula gives the scientific efficiency $E = q(H + h) \div (Q + q)H$.

The following results were obtained by Christopher and Stramberg at the University of Washington in tests of a Hill hydraulic ram, operating under a head of 50 ft., through 140 ft. of 10-in. drive pipe:

Experiment number		20	24	32
Strokes per minute		78	72	51
Water supply, cubic feet per second	$= Q + q$	1.073	1.372	2.418
Water wasted, cubic feet per second	$= Q$	0.657	1.096	2.136
Water pumped, cubic feet per second	$= q$	0.416	0.276	0.282
Supply head, feet	$= H$	50.450	50.300	49.400
Pumped head, feet	$= h$	68.700	167.350	278.150
Delivery head, feet	$= H + h$	119.150	217.650	327.550
Efficiency, Rankine, per cent		86.200	83.800	74.400
Efficiency, D'Aubisson, per cent		91.200	87.000	77.200

PUMP DETAILS

Valve Motions.—The principal features of all engine valve motions are that the valve moves while the piston is standing still (as in reversing) and *vice versa*. In a single direct-acting pump this is effected by having the main valve operated by an auxiliary steam cylinder with piston to which the valve is attached. A small pilot valve controlling admission to and exhaust from this cylinder is mechanically operated by the main steam piston. An example of such a valve motion is the Cameron pump, illustrated in Fig. 16. The four elements mentioned can always be recognized, although they may be combined in many ways. The Worthington duplex pump valve motion presents the same elements, but here both pistons are performing work (Fig. 2).

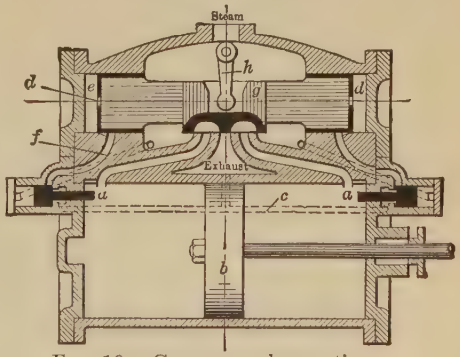


FIG. 16.—Cameron valve motion.

Crank and flywheel pumping engines equipped with Corliss releasing gear are used extensively for all services. The usual reason for using a Corliss gear is the convenient control by the fly-ball governor. This is not so necessary in a pumping engine, where the load is constant and where great sensitiveness is not needed nor desired. A fixed cutoff and hand throttle adjustment is most satisfactory in a pump which acts as a regulator in itself between certain limits.

Water Ends.—Small pumps are generally of the submerged piston or plunger type. The cap or force chamber is removable, also the discharge valve plate

for access to the suction valves. Larger pumps are provided with hand holes. General service direct-acting pumps (150 lb. pressure) up to size 10 by 6 by 10 in. and low service pumps (75 lb. pressure) up to 16-in. piston are built of the submerged type.

General service pumps, 8-in. diameter and over, are generally of the straight-way type, Fig. 17. The suction valves are below the plunger, the discharge valves above. In this way a more nearly straight flow of water through the pump is provided for. They are built as plunger and ring pumps or as piston and sleeve pumps.

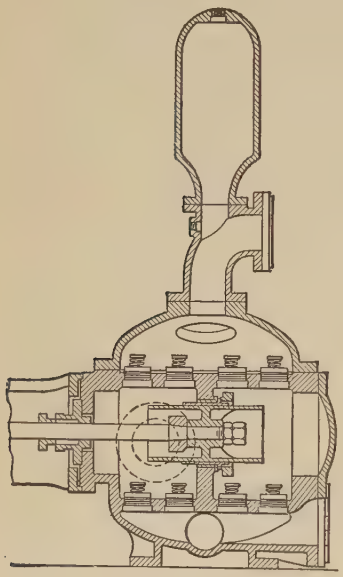


Fig. 17.—Straightway plunger and ring pump.

The best design of a pump for a pressure up to 300 lb. is the outside center-packed pump, illustrated in Fig. 18. The plunger can be watched and any cutting readily detected from the outside. Leakage can be stopped while the pump is in operation.

Pressure Pumps.—For high pressures the pump is sectionalized, by casting the barrel separate from the valve chambers. This construction also facilitates the replacement of broken parts. For very high pressures the valves are placed in individual pots carrying one, two or four valves.

Packing.—Two methods are used to prevent leakage through a working fit. One is to fit the two rubbing parts together closely, depending upon the fit to make the joint. This is used in a plunger and ring pump, where these two parts are ground, so as to leave a clearance of not over 0.001 in. The slight leakage through this clearance is insignificant and will remain so for a long time. Besides the pump is always ready to start even after long idleness. The other method is to insert

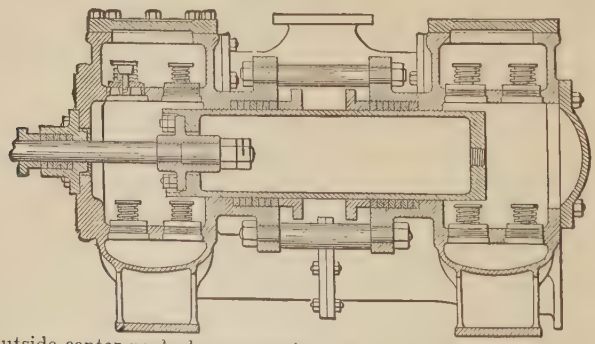


Fig. 18.—Outside center-packed pump, twin pattern and rod attached to end of plunger.

some elastic material which by the pressure against the two surfaces will prevent leakage. This material may be some flexible material, such as hemp, cotton, leather, etc. or it may be made of sections of some suitable metal, usually set out

by springs. The rod stuffing boxes, water pistons, inside packed plungers, and outside center- or end-packed plungers are generally packed with flexible packing. Pump pistons for oil and for many other substances are fitted with metallic packing rings. Metallic packing is rarely used on the pump rods or plungers on account of the impossibility of lubricating them. The plungers in this case should be chilled iron to withstand the effect of the metallic packing and the water should be clear and free from grit.

It is difficult to pack a pump piston so as to be tight without causing undue friction. Even then the tightness will rapidly wear away and the leakage or slip will increase unless constant attention is paid to it.

Pump Valves.—Small disc valves, made of rubber, brass or leather-faced material, are extensively used, Figs. 19, 20 and 21. Discs 4½ in. in diameter are believed to be the most suitable, considering quiet working and initial cost. The valve seat is screwed into the valve deck, and the stem into the seat on a fine taper. The rubber composition is selected so as to be most suitable for the service. For wet vacuum pumps soft rubber is used, for low service (75 lb.) medium soft, for general service (150 lb.) medium hard, for a pressure of 300 lb. hard rubber, but for pressures over 125 lb. metal valve discs are preferable to rubber. Hot water demands a special composition. Thickness of rubber valve disc $t = 0.05s\sqrt{p}$; where s = span, inches, p = water pressure, pounds per square inch.

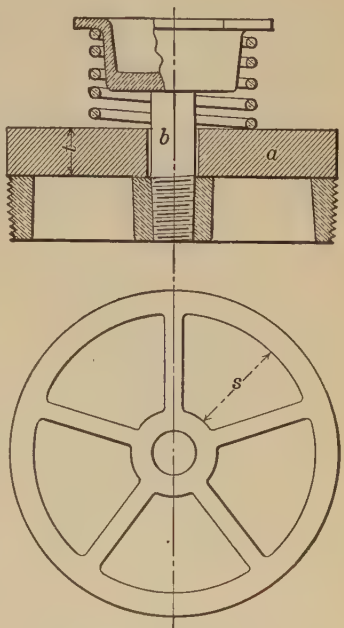


Fig. 19.—Rubber disc valve.

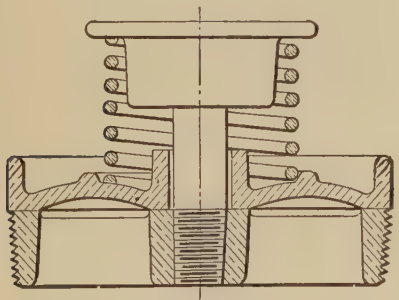


Fig. 20.—Brass disc valve.

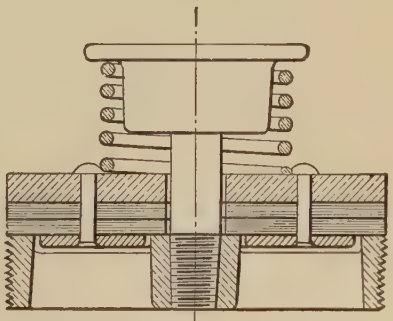


Fig. 21.—Valves faced with leather.

The valve area is the free area through the valve seat, and equals the area of the inner circle, Fig. 19, minus the area of the ribs and hub. The water velocity through this valve area should not be over 222 ft. per minute, which requires a valve area of 45 per cent of the plunger area at 100 ft. piston speed. Frequently 50 and 60 per cent are specified, corresponding to velocities of 200 and 167 ft. per minute, respepervely

The surface of the valve seat is determined by the strength of the material. $F = pA/b$, where F = surface of seat, square inches, p = pressure on the back of the valve, pounds per square inch; A = area of the valve on which p is acting, square inches, and b = bearing pressure of valve on seat, pounds per square inch = 10,000 lb. for steel, 2,000 for red brass, bronze or gun metal, 2,500–3,000 for phosphor bronze, 1,000 for cast iron, 750 for leather and 375 for rubber. For high-speed pumps these values should be reduced to allow for shocks.

The spring pressure produced under the valve, which is the spring load in pounds, divided by the area of the valve A, may be 1 per cent of the working pressure, for the discharge valve, with a maximum of 5 lb. per square inch. For the suction valve the spring pressure should be limited to 0.5 lb. per square inch. If the suction comes in under a head heavier springs should be used.

Figure 22 shows a leather disc valve, Fig. 23 a conical wing valve, $h_1 = h \cos a$, where h_1 = lift of valve, h = free opening measured perpendicular to the seat, a =

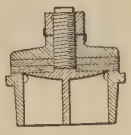


Fig. 22.—Leather-faced wing valve.

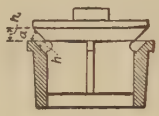


Fig. 23.—Conical wing valve.

angle of valve; for $a = 45°$, $h_1 = 1.41h$, or valve must lift 41 per cent higher than a flat-seated valve to give the same opening. Figure 24 shows a clapper valve, Fig. 25 a leather-faced clapper valve and Fig. 26 a ball valve, used for thick liquids.

Air Chambers.—The object of an air chamber is to provide an elastic element in the pipe line and produce a uniform flow. Simple pumps require large air cham-

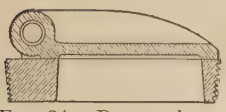


Fig. 24.—Bronze clapper valve.

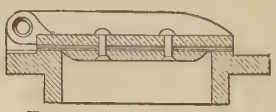


Fig. 25.—Leather-faced clapper valve.

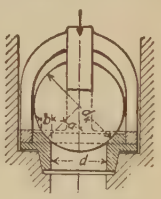


Fig. 26.—Ball valve.

bers (eight times the displacement of the plunger per stroke), duplex pumps only small air chambers (four times the displacement), duplex low service (75 lb. pressure) and small general service pumps require no air chamber. They are a necessity on all crank and flywheel and power pumps. If the air coming in with the water is not sufficient to replenish that absorbed by the discharge water an air charging device should be used. This may be an independent air compressor or simply a pipe connected to one of the pump chambers, and provided with two check valves for air inlet and outlet. On pumps for over 300-lb. pressure, where the air is absorbed rapidly by the water a spring-loaded ram, called an alleviator should be used, Fig. 27.

A vacuum chamber performs the same duty on the suction pipe and should have its inlet on the dead end of a tee so as to be in line with the current, Fig. 28. It should be placed on the suction pipe of every good-sized pump and is a necessity on high lifts and pumps producing an irregular flow. When the water carries much air the vacuum chamber may act as an air collector from

which the air may be removed by a vacuum pump. In long suction pipes where the air has more time to become liberated from the water the chamber may be constructed as a separating box.

The object of a foot valve is to keep the pump primed during a stoppage. It adds to the friction and reduces the possible suction lift. Figure 29 shows a clack valve and strainer. Figure 30 shows one fitted with a number of small disc valves. If the water carries impurities that might do damage to the pump a strainer should be provided in the suction pipe.

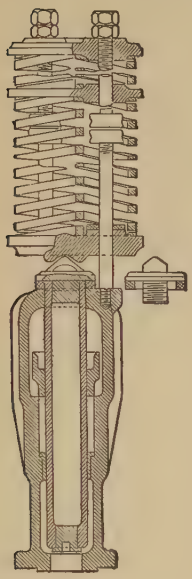


FIG. 27.—Alleviator.

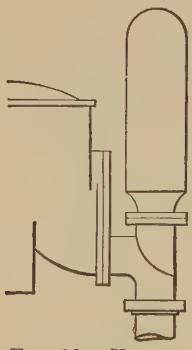


FIG. 28.—Vacuum chamber.

FIG. 29.—Clack valve and strainer.

The ideal suction lift of a pump equals the water column sustained by the atmospheric pressure minus the vapor pressure; the former is affected by the barometer and altitude, the latter by the temperature of the water. The hydraulic losses, which comprise the friction through the suction pipe, valves and

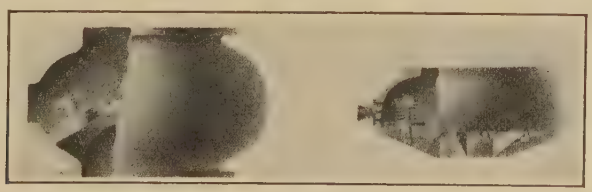


FIG. 30.—Foot valves.

passages, and the velocity head are generally insignificant as compared with the influence of the air contained in the suction water, the effect of which can not be computed. Figure 31 is an empirical diagram, giving the ideal suction lift t, the attainable suction lift a for sea level, the suction head required to make the water flow into the pump for high temperatures and the maximum possible suction lift under favorable conditions m, *i.e.*, when the suction pipe is tight and the water does not carry an excessive amount of air. Curves a and m are based on a velocity of 3 ft. per second and a short suction pipe with one bend. The suction lift is measured vertically from the water in the well to the underside of the

discharge valves and the suction head from there to the water level in the supply tank. A suction head of 12 ft. is sufficient for any temperature at the sea level, because water can not exist at a temperature above 212°F. except under pressure, the effect of which is added to the static head. The broken lines parallel to *a* indicate the lifts attainable or heads required at various altitudes.

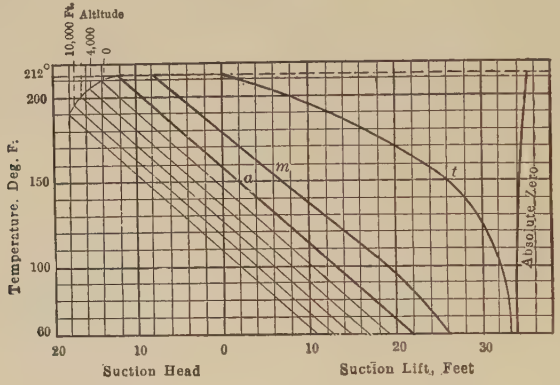


Fig. 31.—Suction lifts and head diagram.

To dispose of the exhaust steam it may be run into the suction pipe where it mixes with the water and is condensed. This method is especially recommended in connection with a mine sinking pump, as it makes the unit self contained. The exhaust pipe is provided with a three-way cock, so that the exhaust can be turned into the atmosphere for starting. A good construction for a suction condenser is to allow the steam to enter an annular space around the suction pipe, to travel upward and enter the suction pipe near the top, where it condenses without causing any disturbance (Fig. 32). The water may be heated thereby 20°.

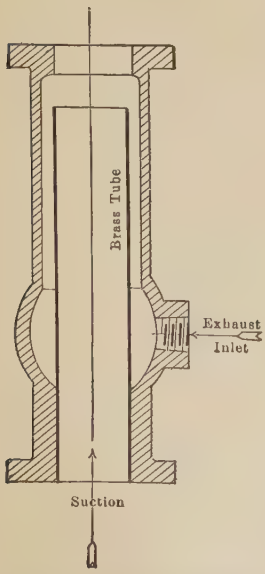


Fig. 32.—Suction condenser.

The exhaust of a pump may also be disposed of by a jet condenser usually equipped with a single direct-acting wet vacuum pump, the object of which is to remove the condensate, the injection water and the air from the tail end of the condenser. The injection water is drawn into the condenser cone from the suction well and is discharged into the sewer by the wet vacuum pump.

By using a properly proportioned condensing pump in place of a non-condensing pump a saving of from 30 to 40 per cent in steam may be effected. Tests investigating the same pump, running condensing and non-condensing, show the gain in power rather than the saving in steam, because in order to develop the same power the steam supply must be cut down when running condensing and the engine is then running under unfavorable conditions for which it was not designed.

Surface condensers have less moving machinery and use less steam for operating than jet condensers, but are liable to be out of service a longer time while the tubes are

being cleaned. A surface condenser may be placed in the suction pipe of the pump, provided there is a place below it for the condensate pump. It is arranged so that the water flows around the tubes, while the exhaust steam flows through the tubes. Only a small wet vacuum pump is then required to remove the condensate and the air. A better place for it is in the discharge pipe as this gives a greater suction head for the vacuum pump. This is much more costly on account of the high pressure for which it must be designed, but an obstruction in the discharge pipe is by far not so objectionable as one in the suction pipe.

Acid Pumps.—When an acid or other corrosive substance is to be handled by a pump no material should be used that is attacked by the substance. For this reason pumps may be constructed either wholly of brass, bronze, rubber or even stone ware, or wholly of iron or they may be merely bronze fitted, which means that certain working parts such as pistons, rods, pump pistons, etc., are made of bronze or bronze lined. In ordering bronze or brass-fitted pumps it should be clearly specified what parts should be made of bronze or brass. In an all-iron pump the pump cylinder is bored to size and has no lining. The valve seats and stems are made of malleable iron.

If the size permits it in a brass fitted pump the piston rod should be split in the cross head, and the steam piston rod should be made of steel. Brass at the high temperature in the steam cylinder is too soft to wear well.

Table 6 gives a very complete list of substances that may be handled by pumps and the materials of which the pump parts should be constructed.

TABLE 6.—MATERIALS FOR PUMP PARTS FOR HANDLING VARIOUS LIQUIDS

Column 2 Condition		Columns 4 and 6 material		Compositions of acid metal					
Con. Dil Ex. c.	Concentrated Diluted Extremely concentrated	AB	All bronze	Cu	Sn	Pb	Sb	Al	Tensile strength
		AI	All iron						
		AM	Acid metal	82	5	12	..	1	
		BF	Brass fitted	88	10	2			
		CI.	Cast iron						
Column 5 valve service		E	Enameled	90	Aluminum Bronze			10	45,000
		HL	Hard lead						
		LB	High lead bronze	75	5	20	Light section		19,000
				80	5	15	Heavy section		15,000
R	Regular valve disc, etc.	MM	Monel metal						
		PB	Phosphor bronze						
L.O.	Large openings No ribs in seat	RF	Regular fitted	½	½ (For mine water)	85½	13½	(Ant. lead)	
	Ball valve	AL	Aluminum						
	Clapper Valve	Cu	Copper						
	Etc.	Pb	Lead						
		Sb	Antimony						
		Sb Pb	Antimonial lead	..	..	90	10	(For acid)	
		Sn	Tin						

The column numbers refer to the columns on pp. 130–134. For handling substances for which material E is specified, pumps made entirely of rubber would be most suitable. The American Hard Rubber Co., New York, N. Y., is developing such a line, which will include single and duplex direct-acting, steam pumps, power and centrifugal pumps, as well as piping, fittings and valves. Pumps handling foods or medicines must be kept clean and should preferably be all bronze.

TABLE 6.—*Continued*

Substance	Condition	Chem. symb.	Material	Valve service	Remarks
Acetic acid	Con.	$C_2H_4O_2$	*E*	*R*	No tin, does not attack C. I.
Acetic acid	Dil.		*E*	*R*	Attacks all common metals
Acid mine water			*LB*	*R*	Wood- or Sb Pb-lined C. I.
Acid mine water			....	*R*	or lined with slag cement. No
Acid mine water			*AM*	*R*	zinc
Alcohol			*AB*	*R*	
Alkaline liquid	Ex. C.		*AI*	*R*	Attacks E more readily than C. I.
Alkaline liquid	Dil.		*E*	*R*	
Alum		$KAl(SO_4)_2$		*R*	Attacks C. I. slowly
Aluminum sulphate		$Al_2(SO_4)_3$	*BF*	*R*	Forms foam
Ammonia		NH_3	*AI*	*R*	Asbestos packing
Ammonium bicarbonate		NH_4HCO_3			Attacks C. I. slowly
Ammonium chloride		NH_4Cl	*AI*	*R*	Is salammoniac
Ammonium nitrate		NH_4NO_3	*AI*	*R*	
Ammonium sulphate		$(NH_4)_2SO_4$	*AI*	*R*	
Aniline water			*AI*	*R*	
Barium chloride		$BaCl_2$	*RF*	*R*	These do not attack C. I.,
Barium nitrate		$Ba(NO_3)_2$	*RF*	*R*	brass or bronze
Beer and beerwort			*AB*	*R*	
Beet juice	Thin		*BF*	*R*	Sugar mill
Benzene, coal tar product		C_6H_6	*AI*	*R*	No rubber
Benzine, oil dist. prod. No. 4		C_nH_{2n+2}	*BF*	*R*	Sp. gr. = 0.74. No rubber
Bichloride of mercury		$HgCl_2$	*AI*	*R*	
Bitterwasser	.....	.....	*AB*	*R*	Hunyadi or similar
Bleachery	.....	.....	*BF*	*R*	
Brine	.....	.....	*AB*	*R*	Refrigeration
Carbolic acid	.....	C_6H_5OH	*AI*	*R*	Disinfectant
Carbon tetrachloride	.....	CCl_4	*RF*	*R*	Is carbona
Carbona	.....	.....	*RF*	*R*	Dissolves grease
Cachaza (mud)	.....	.....	*BF*	*LO*	Sugar mill
Calcium acid sulphate	Con.	$CaSO_4$	*AB*	*R*	
Calcium acid sulphate	Dil.		*BF*	*R*	
Calcium chlorate		$Ca(ClO_3)_2$	....	*R*	Cu is best, then : AB, C. I.
Calcium hypochlorite	.....	$Ca(OCl)_2$	*E*	*LO*	Disinfecting agent
Calcium chloride brine	.....	$CaCl_2$	*BF*	*R*	Refrigeration,
Calcium magn. sod. chloride	.....	.....	*BF*	*R*	Salt-well brine
Carbonate of soda	.....	Na_2CO_3	*AI*	*R*	Is sal soda
Carbonic acid gas in water	.....	CO_2	*RF*	*R*	
Caustic sulphide	KSH	or NaSH	*AB*	*R*	
Caustic soda	.....	NaOH	*AI*	*R*	No iron if NaCl is present
Caustic lye, or potash	.....	KOH	*AI*	*R*	
Caustic manganese	.....	$Mn(OH)_2$	*AI*	*R*	Is unstable compound, oxidizes in air to MnO·OH
Caustic strontia (strontium salt)	.....	$Sr(OH)_2$	*AB*	*LO*	Heat by steam to prevent crystallization
Chloride of lime	.....	$CaOCl_2$	*E*	*LO*	Bleaching powder, disinf. agent
Chloride of zinc	.....	.....	*AI*	*R*	
Chlorine and water	.....	Cl	*E*	*R*	Cu or Ni, lubricate w. 60°Bé. H_2SO_4
Caustic zinc chloride	.....	ZnClOH	*BF*	*R*	Deposits $Zn(OH)_2$, zinc hydroxide
Citric acid	.....	.....	*RF*	*R*	
Coal tar oil	.....	.....	*AI*	*R*	
Copperas	.....	$FeSO_4$	*AI*	*LO*	Is green vitriol
Copper nitrate	....	$Cu(NO_3)_2$	....	....	Cu is best. Attacks Fe and Pb vigorously

TABLE 6.—*Continued*

Substance	Condition	Chem. symb.	Material	Valve service	Remarks
Copper sulphate	.....	$CuSO_4$	*AB*	*R*	Is blue vitriol
Creosote	.....	.....	*BF*	*R*	No rubber
Creosote oil	.....	.....	*AI*	*R*	No rubber
Cyanide	.....	NaCN	*AI*	*R*	Chilled or high silicon iron, or wood lined C. I.
Cyanide of potassium	.....	KCN	*AI*	*R*	Attacks Au, Ag, Cu, Fe, Pb, Zn, high silicon iron
Cyanic acid	.....	CNOH	*AI*	*R*	
Cyanogen (gas)	.....	CN	*AI*	*R*	Excessively poisonous
Cellulose	.....	.....	*RF*	*LO*	
Dish water	.....	.....	*RF*	*R*	
Distillery wort	.....	.....	*AB*	*R*	
Dye wood liquor	.....	.....	*BF*	*R*	
Fatty acid, ...over	55°C.	.....	*AB*	*R*	Stearic, cerotic, palmitic
Ferric hydroxide	.....	$Fe(OH)_3$	*AB*	*R*	
Ferrous chloride	.....	$FeCl_2$	*Eb*	*R*	Ebonite (rubber treated with chlorine)
Ferrous sulphate	.....	$FeSO_4$	*AI*	*LO*	Is green vitriol
Gasoline, oil dist. prod. No. 2.	.....	C_nH_{2n+2}	*BF*	*R*	No rubber. Sp. gr. = 0.66
Glue	Hot		*BF*	*LO*	
Glycerine	.....	$C_3H_5(OH)_3$	*AB*	*R*	Sp. Gr. = 1.26
Gun cotton brine	.....	.....	*AB*	*R*	
Grape juice	.....	.....	*AB*	*R*	No iron. Wood in contact with G. J. develops alcohol
Heptane	.....	.....	*BF*	*R*	Is naphtha
Hops	.....	.....	*BF*	*R*	
Hydrochloric acid	.....	HCl	*E*	*R*	Is muriatic acid
Hydrocyanic acid	.....	HCN	*AI*	*R*	
Hydrofluosilic acid	.....	H_2SiF_6	*AB*	*R*	
Hydrosulphite	.....	HSO_2	Sb Pb	*R*	Sb Pb lined C. I.
Hyposulphite of soda	.....	$Na_2S_2O_3$	Sb Pb	*R*	Iron piston in Sb Pb lining pist. ring: 8 Cu 1 Sn, no zinc
Kerosene, oil dist. prod. No. 5.	.....	.....	*RF*	*R*	No rubber. Sp. gr. = 0.8
Lard	Hot	.....	*BF*	*R*	No rubber
Lead nitrate	.....	$Pb(NO_3)_2$			
Lime water	.....	$Ca(OH)_2$	*AI*	*LO*	
Linseed oil	.....	.....	*AI*	*LO*	No rubber
Lye, salty	.....	.....	*BF*	*R*	Containing much salt
Magnesium chloride	Hot	$MgCl_2$	*HL*	*R*	Hydrolyzes, liberating HCl
Magn. acid sulphate	Con.	.....	*AB*	*R*	
Magn. acid sulphate	Dil.	.....	*BF*	*R*	
Magn. oxychloride	.....	MgClOH	*HL*	*R*	Is basic magn. chloride deposits insol. salt
Magnesium sulphate	.....	$MgSO_4$	*AI*	*R*	Is epsom salts
Magma	.....	.....	*BF*	*LO*	Is thick residue, use suc. valveless, cen. inlet pump
Marsh gas	.....	CH_4	*RF*	*R*	
Mash	.....	.....	*BF*	*LO*	
Milk	.....	.....	*AB*	*R*	
Milk of lime	.....	$Ca(OH)_2$	*AI*	*LO*	
Mine water	.....	.....	....	....	See acid mine water
Molasses	.....	.....	*BF*	*LO*	If very heavy use Magma P_o
Muriatic acid	.....	HCl	*E*	*R*	

TABLE 6.—*Continued*

Substance	Condition	Chem. symb.	Material	Valve service	Remarks
Naphtha, oil dist. prod. No. 3.	.....	C_nH_{2n+2}	*BF*	*R*	No rubber. Sp. gr. =0 .7
Nickel chloride	.....	$NiCl_2$	*Cu*		
Nickel sulphate	.....	$NiSO_4$	*Cu*		
Nitric acid	Con.	HNO_3	*Pb*	*R*	No rubber
Nitric acid	Dil.	.....	*AI*	*R*	No rubber
Oil, crude	.....	.....	*RF*	....	Large valve area No rubber
Oil, heavy lubr. oil dist. prod. No. 6	.....	.....	*BF*	....	No rubber. Sp. gr = 0.9
Oil, mineral	.....	.....	*RF*	*R*	No rubber
Oil, vegetable	.....	.....	*AI*	*R*	No rubber
Paraffine	Hot	C_nH_{2n}	*BF*	*R*	Narrow valve face
Peroxide of hydrogen	.....	H_2O_2	*AB*	*R*	Corrodes iron
Petroleum	.....	.....	*RF*	*R*	No rubber
Petroleum ether	.....	.....	*RF*	*R*	Is gasoline
Potash	.....	K_2CO_3	*AI*	*R*	
Potassium alum	.....	$Al_2K_2(SO_4)_4$	....	*R*	Attacks C. I. slowly
Potassium carbonate	.....	K_2CO_3			
Potassium chloride	.....	KCl	*AB*	*R*	Corrodes iron
Potassium cyanide	.....	KCN	*AI*	*R*	See cyan. of pot.
Potassium nitrate	.....	KNO_3	*AB*	*R*	Bronze is best, then brass, then C. I., the sulphate does not react with Pb or H.L., the nitrate does.
Potassium sulphate	.....	K_2SO_4	*AB*	*R*	
Potash sulphide	.....	K_2S	*AI*	*R*	
Rapeseed oil	.....	.....	*AI*	*R*	
Rectifying pump	.....	.....	*AB*	*R*	Distillery
Residuum	.....	.....	*BF*	*LO*	
Rhigolene, oil dist. prod. No. 1	.....	.....	*BF*	*R*	No rubber. Sp. gr. = 0.62
Salammoniac	.....	NH_4Cl	*AI*	*R*	
Salt brine 3 per cent	Salt	NaCl	*BF*	*R*	Salt water attacks C. I. and liberates the graphite
Salt brine over 3 per cent	Salt	.....	*AB*	*R*	
Sea water	.....	.....	*BF*	*R*	
Sewage	.....	.....	*BF*	*LO*	
Slop	.....	.....	*BF*	*LO*	Brewery
Soap liquor	Thin	.....	*AI*	*LO*	Alkali
Soap nigger	.....	.....	*AI*	*LO*	
Soap water	.....	.....	*AI*	*R*	
Soda	.....	NaOH	*AI*	*R*	
Soda ash	.....	.....	*AI*	*R*	
Sodium bicarbonate	.....	$NaHCO_3$	*AI*	*R*	Is baking powder and forms foam
Sodium brine	.....	.....	*BF*	*R*	
Sodium carbonate	.....	Na_2CO_3	*AI*	*R*	Is sal soda
Sodium chloride	.....	NaCl	*AB*	*R*	Is common salt
Sodium hydroxide	.....	NaOH			
Sodium hyposulphite	.....	NaS_2O_3	*AI*	*LO*	
Sodium nitrate	.....	$NaNO_3$			
Sodium sulphate	.....	Na_2SO_4	*AI*	*LO*	
Starch	.....	.....	*BF*	*LO*	
Strontium nitrate	.....	$Sr(NO_3)_2$	*AB*	*R*	
Sugar	.....	.....	*AB*	*LO*	
Sulphate of lime	.....	$CaSO_4$	*AB*	*LO*	
Sulphide of hydrogen	.....	H_2S	*RF*	*R*	Acid
Sulphide of sodium	Hot	Na_2S	*AI*	*R*	

TABLE 6.—*Concluded*

Substance	Condition	Chem. symb.	Material	Valve service	Remarks
Sulphide of sodium	Cold		*RF*	*R*	
Sulpholignic salts	Con.		*AB*	*R*	
Sulpholignic salts	Dil.		*BF*	*R*	
Sulphur dioxide		SO_2	AB	*R*	Refrigeration
Sulphuric acid	Con.	H_2SO_4	*AI*	*LO*	Is oil of vitriol. Sb Pb fitted
Sulphuric acid	Dil.	.....	*HL*	*LO*	Wood- or Sb Pb -lined C. I.
Sulphuric acid	Dil.	.....	*AB*	*LO*	No zinc
Sulphuric acid, fuming	.....	$H_2S_2O_7$	*RF*	*LO*	
Sulphuric acid, 60°Bé	Hot	.....	*AI*	*R*	
Sulphuric acid, 40°Bé	Hot	.....	*Pb*	*R*	
Sulphurous acid	Con.	H_2SO_3	*E*	*R*	
Sulphurous acid	Dil.	.....	*AB*	*R*	
Sulphurous acid, gaseous	.....	.....	*BF*	*R*	
Sweet water	.....	.....	*BF*	*R*	Sugar mill
Syrup	.....	.....	*AB*	*LO*	
Tan liquor	.....	.....	*AB*	*R*	
Tar	.....	.....	*AI*	*LO*	
Tar and ammonia in water	.....	.....	*AI*	*R*	Traces
Turpentine oil	.....	.....	*AI*	*R*	No rubber
Urine	.....	.....	*AB*	*R*	
Vinegar	.....	.....	*AB*	*R*	Is impure diluted acetic acid. Affects Fe less than bronze
or	.....	.....	*AM*	*R*	
Vitriol, blue	.....	$CuSO_4$	*AB*	*R*	
Vitriol, green	.....	$FeSO_4$	*AI*	*LO*	
Whiskey	.....	.....	*AB*	*R*	
Wine	.....	.....	*AB*	*R*	Usually contains malic, acetic or tartaric acid
Wood pulp	.....	.....	*BF*	*LO*	Paper mill
Wort	.....	.....	*AB*	*R*	Is infusion of malt
Yeast	.....	.....	*BF*	*LO*	
Zinc nitrate	.....	$Zn(NO_3)_2$	*AB*	*R*	Like potassium nitrate and sulphate
Zinc sulphate	.....	$ZnSO_4$	*AB*	*R*	

The capacity of a pump is generally understood to mean the actual amount of water pumped per unit of time. $Q = VE_v$, where Q = capacity, usually in gallons per minute, or 24 hr., V = net displacement of plunger or pump piston in same unit, E_v = volumetric efficiency = 1 − slip. The net displacement V = displacement of plunger minus displacement of rod. In an ordinary case, where the rod is submerged at one end only, it acts like a single-acting plunger, and in opposition to the main plunger. Displacement of plunger $V_p = (D^2\pi \div 4)S \times 12 \div 231 = 0.0408D^2S$ U. S. gal. per minute. Displacement of rod $V_r = 0.0204d^2S$, where D = diameter of plunger, inches, and d = diameter of the plunger rod, inches, and S = piston speed feet per minute, then $V = V_p - V_r$.

From Table VII the displacement, V_p, of a single double-acting plunger can be found for a piston speed of 100 ft. per minute. To find the net displacement deduct the displacement of the rod, V_r, which if submerged at one end only, is one-half of the value given in the table.

For any other piston speed multiply the result by $S/100$.

For single single-acting pumps divide V by 2, for duplex double-acting pumps multiply V by 2, for triplex single-acting pumps multiply by $3/2$, etc. In some types, such as outside end-packed plunger pumps the rod is not submerged and the full area of the

plunger is effective. For a first trial assume S = 100 ft. per minute, and allow 6 per cent for slip and rod. With this compute the approximate diameter of the plunger, select a suitable length of stroke and recalculate with correct piston speed derived from Table 1.

TABLE VII.—DISPLACEMENT OF A SINGLE DOUBLE-ACTING PLUNGER

Diameter of plunger, inches	Displacement based on 100 feet piston speed		
	Gallons per minute	Gallons per 24 hours	Cubic feet per second
½	1.02	1469	0.002273
⅝	1.594	2295	0.003552
¾	2.295	3305	0.005114
⅞	3.124	4499	0.006061
1	4.080	5875	0.009091
1¼	6.375	9180	0.014205
1½	9.180	13,220	0.020455
1¾	12.495	17,992	0.027840
2	16.320	23,501	0.036364
2¼	20.655	29,744	0.046020
2½	25.500	26,720	0.056820
2¾	30.850	44,425	0.068740
3	36.72	52,880	0.0818
3½	49.98	71,980	0.1114
4	65.28	94,010	0.1455
4½	82.62	118,970	0.1841
5	102.00	146,900	0.2273
5½	123.45	177,800	0.2757
6	146.90	211,500	0.3273
6½	172.40	248,250	0.3841
7	199.90	287,850	0.4454
7½	229.50	330,500	0.5114
8	261.10	373,700	0.5818
8½	294.80	424,500	0.6570
9	330.50	475,900	0.7364
9½	368.30	530,200	0.8207
10	408.0	587,500	0.909
11	493.7	711,000	1.100
12	587.5	860,400	1.309
13	689.5	992,900	1.536
14	799.7	1,151,500	1.782
15	918.0	1,322,000	2.045
16	1,044.5	1,504,000	2.322
17	1,179.0	1,698,000	2.627
18	1,322.0	1,903,500	2.946
19	1,473.0	2,121,000	3.282
20	1,632.0	2,348,000	3.636

The speed of a pump may be stated as rotative speed in revolutions or cycles per minute, or as piston speed in feet per minute. $S = sn/6$, where S = piston speed, feet per minute; n = revolutions per minute; s = stroke in inches. In direct-acting pumps one revolution means one complete cycle of any reciprocating part. Table I gives the proper speeds of all types of reciprocating pumps.

The mechanical efficiency of a pump is the power utilized in percentage of the power expended. E_m = Pump horsepower/Indicated horsepower. Or E_m = Pump horsepower ÷ Brake horsepower, in the case of a power pump. It accounts for all mechanical losses in transmitting the power from the power end to the pump end. Table VIII gives usual E_m for various types of pumps and different lengths of stroke.

TABLE VIII.—MECHANICAL EFFICIENCIES IN PER CENT

Stroke, inches	Direct-acting Pumps				Direct acting water motors	Crank and fly-wheel pumps
	Plunger pumps and piston pumps	Outside packed pumps up to 300 pounds per sq. inch	Pressure Pumps			
			up to 1,000 pounds per sq. inch	up to 3,000 pounds per sq. inch		
3	50.0	47	45	39	31	
4	55.0	52	50	43	34	
5	60.0	57	54	47	37	
6	65.0	61	58	51	40	
8	70.0	66	63	55	43	
10	75.0	71	67	58	46	
12	77.5	74	70	60	48	80.0
15	80.0	76	72	62	49	82.5
18	82.5	78	74	64	51	85.0
24	85.0	81	77	66	52	87.5
30	85.0	81	77	66	52	87.5
36	87.5	83	79	68	54	90.0
48	90.0	85	81	70	55	92.5
60	90.0	85	81	70	55	92.5

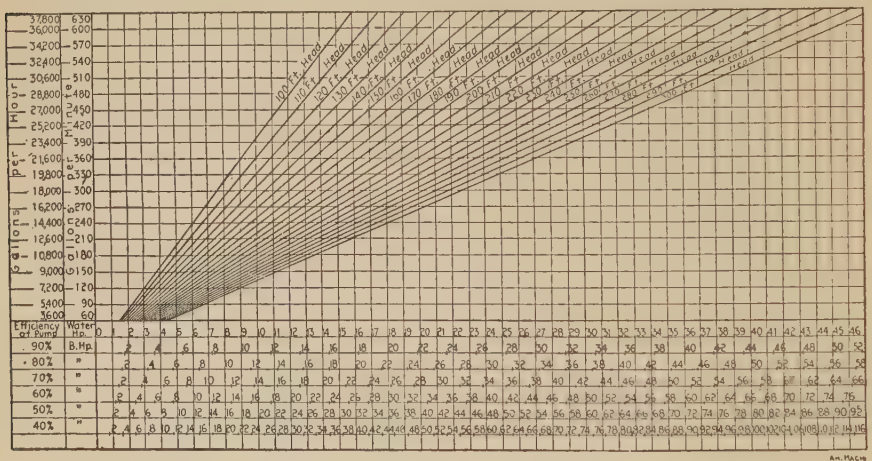
The Volumetric Efficiency of a pump is the capacity in percentage of the displacement. $E_v = Q/V$. This efficiency gives information as to the losses by leakage. $E_v = 1 - \text{slip}$.

The slip, or the water leakage past the plunger and through the valves, in a pump in good condition, running at normal speed, is not over 3 per cent. It is approximately inversely proportional to the speed and at very slow speed may be 100 per cent. This speed, for which the capacity = 0, is of value in testing small pumps; it is expressed by the number of strokes the pump makes with the discharge valve closed and with sufficient pressure on the steam piston to produce the normal water pressure. It is called the lost actions, and should not be more than 1 per cent of the normal speed. The proportion of lost actions to normal speed gives a measure of the slip. The leakage past the plunger may be eliminated in an outside-packed plunger pump by properly packing the stuffing boxes and is very little in a well made plunger and ring

pump. A piston, if newly packed may be perfectly tight, but will in time wear and exhibit signs of leakage. Depending upon the conditions of the valves and the speed the leakage may have any value between 0 and 100 per cent.

The hydraulic efficiency is the total head realized in percentage of the head produced in the pump. $E_h = H \div H_i$, where H = total head pumped against (static head + friction + velocity head), and $H_i = H +$ hydraulic losses in the pump. It gives information as to all losses due to the passing of the water through the pump, from the lower water level to the discharge nozzle. In a reciprocating pump where the velocity is low these losses are small and are difficult to determine. The hydraulic efficiency is, therefore, generally included in the term "mechanical efficiency" in which case E_m = water horsepower ÷ indicated horsepower or = water horsepower ÷ brake horsepower.

Thermal efficiency, $E_t = 2{,}546\ hp./S(H - h)$, where 2,546 = heat equivalent of 1 hp. in B.t.u. per hour; S = steam consumed, pounds per hour; H = total heat in 1 lb. of steam at initial pressure; h = total heat in 1 lb. of feed water.



PUMPING DIAGRAM, 100- TO 300-FT. HEAD (U. S. GALLONS)

The total efficiency is the product of the mechanical, the volumetric and the hydraulic efficiencies. $E = E_m \times E_v \times E_h$.

Overall efficiency (in an electrically driven pump) E_o = water horsepower ÷ electrical horsepower. It is the ratio of the actual power developed in the water end, *i.e.*, pounds of water raised times equivalent head ÷ 33,000, to the amount of electrical energy delivered to the motor as measured appropriately on the motor switchboard.

The water horsepower is the effective work done by the pump in lifting and transporting the water. It is equal to the pounds of water delivered per minute multiplied by the difference in elevation between the water level in the suction well and that in the reservoir plus the friction in the pipe divided by 33,000. $HP_w = WH \div 33{,}000$, where W = weight in pounds of water pumped = Gw, where G = amount of liquid actually discharged in United States gallons per minute, w = weight of 1 gal. (= 8.33 lb. for water at 62°F.); H = total head in feet against which the pump operates (generally measured by a gage in the water column to which reading is added the vertical distance between the level in the section well and the center of the gage. 1 lb. per square inch gage = 2.31 ft. head at 62°F.

The pump horsepower is the total work done by the pump, $HP_p = HP_w -$

hydraulic losses in pump = HP_w/E_h. It is the indicated pump horsepower, but is difficultly determined on account of the unreliability of pump indicator cards.

The brake horsepower, in the case of a power pump, is the horsepower delivered to the pump, as measured by a dynamometer or suitable means. Figure 33 gives the water horsepower and the brake horsepower required at usual quantities and heads. For other conditions multiply or divide the horsepower obtained from the diagram proportionately. $HP_b = HP_w/E_m$(E_h included in E_m).

Example.—The theoretical water horsepower required to pump 500 gal. per minute against 300 ft. head is, as calculated from the indicator diagram, 38.5 horsepower, at 80 per cent efficiency the brake horsepower required will be 48 hp.

The indicated horsepower, in the case of a steam pump $HP_i = HP_w/E_m$. It is determined from indicator cards taken off the steam end.

The input horsepower, in the case of an electrically driven pump, or the electric horsepower, is the horsepower consumed by the motor, measured at its switchboard. In the case of a steam-or air-driven pump, the input horsepower may be based on the steam or air consumption, computed at appropriate quantities per horsepower.

Calculation of the Steam End.—In computing the size of a crank and flywheel pumping engine it is best to find the required indicated horsepower and then proceed in the usual manner of calculating a steam engine. The reason for this is that usually the steam end consists of a single unit, while the pump end may be composed of a number of pumps, either single- or double-acting. The size may be figured closely, because any lack of power may be made up by a later cutoff. Non-condensing sizes should be figured too small rather than too large, so that there will be no danger of expanding below the atmospheric line.

In a direct-acting steam pump the horsepower can be ignored and the steam piston force F balanced against the plunger load L + the mechanical friction. $L = F \times E_m$, where L = area of plunger × total water pressure; F = area of steam cylinder × mean effective pressure. In the case of a multi-stage steam end, the mean effective pressure, called p_e hereafter, is best referred to the area of the high-pressure cylinder and can be calculated by the following formulas, in which p_i = absolute initial pressure in high pressure cylinder; b = back pressure; R = ratio of cylinder areas in a compound pump; and R and R_1 = the ratios of cylinder areas in a triple expansion pump, R = low pressure area/high pressure area and R_1 = low pressure area/high pressure area. Simple steam cylinder direct-acting pumps (so-called high-pressure pumps); $p_e = p_i - b$. Compound pumps: $p_e = 2p_i - (p_i/R) - bR$; $R = \sqrt{p/b}$. Triple expansion pumps: $p_e = 3p_i - (2p_i/\sqrt{R}) - bR_1$; $R_1 = \sqrt[3]{(p/b)^2}$ and $R_1 = R^2$. In these formulas the following values may be substituted: p_i = gage pressure at throttle + 10 lb.; b = 16 lb. for non-condensing, 5 lb. for condensing compound pumps and 4 lb. for condensing triple expansion pumps. $R \leq 4$ for compound pumps and $R_1 = \frac{\text{low pressure area}}{\text{high pressure area}} = \leq 8$ for triple-expansion. These values cover unfavorable conditions, such as direct-acting pumps must be designed for.

The duty of a pump is its performance based on the output in foot-pounds per 1,000 lb. of dry steam, or per 100 cu. ft. of air, or per 1,000,000 B.t.u. furnished by the boilers at appropriate pressure. The duty of a steam pump can be calculated approximately from the indicator diagram (either actual or ideal) by the formula: card duty = 144,000 pv, and duty = card duty/$E_m E_s$ where p = mean effective pressure of whole steam end referred to the area of any one cylinder, v = specific volume of steam at the terminal pressure in this same cylinder, E_m = mechanical efficiency, and E_s = Steam efficiency, comprising all steam losses.

Data and Results of Duty Trial of Steam Pumping Machinery

(A. S. M. E. Code of 1915)

(For a short report the items designated by letters of the alphabet may be omitted.)

(1) Test of.....pump located at.....
To determine.....Test conducted by.....

Dimensions, Etc.

(2) Type of machinery.....
(3) Rated capacity in gallons per 24 hr.....gal.
(4) Size of engine or turbine.....(5) Size of pump.....
(6) Auxiliaries (steam- or electric-driven).....
(*a*) Type and make of condenser equipment.....
(*b*) Rated capacity of condenser equipment.....
(*c*) Type of oil pump, jacket pump, and reheater pump (direct or independently driven).....

Date and Duration

(7) Date.....(8) Duration.....hr.

Average Pressures and Temperatures

(9) Pressure in steam pipe near throttle by gage.....lb.
(10) Barometric pressure.....in.
(*a*) Steam chest pressure.....lb.
(*b*) Pressure in receivers and reheaters by gage.....lb.
(*c*) Pressure in turbine stages by gage.....lb.
(11) Pressure in exhaust pipe near engine or turbine by gage.....lb.
(12) Vacuum in condenser.....in.
(*a*) Corresponding absolute pressure.....lb.
(*b*) Absolute pressure in exhaust chamber.....lb.
(13) Temperature of steam, if superheated, at throttle.....deg.
(*a*) Normal temperature of saturated steam at throttle pressure.....deg
(*b*) Temperature of steam leaving receivers, if superheated.....deg.
(14) Temperature of steam in exhaust pipe near engine or turbine.....deg.
(*a*) Temperature of circulating water entering condenser.....deg.
(*b*) Temperature of circulating water leaving condenser.....deg.
(15) Pressure in force main by gage.....lb.
(16) Vacuum or pressure in suction main by gage.....in. or lb.
(*a*) Correction for difference in elevation of the two gages.....lb.
(17) Total head expressed in pounds pressure per square inch.....lb.
(*a*) Total head expressed in feet.....ft.

Quality of Steam

(18) Percentage of moisture in steam near throttle, or number of degrees of superheating.....per cent or deg.

Total Quantities

(19) Total water fed to boilers.....lb.
(20) Total condensed steam from surface condenser (corrected for condenser leakage).....lb.
(21) Total dry steam consumed (Item 19 or 20 less moisture in steam).....lb.
(22) Total gallons of water discharged, by measurement.....gal.
(*a*) Total gallons of water discharged, by plunger displacement, uncorrected.....gal.

(*b*) Percentage of slip [100 (Item 22*a*-Item 22)/Item 22*a*.] .....per cent
(*c*) Leakage of pump[1].....gal.
(*d*) Total gallons of water discharged, by calculation from plunger displacement, corrected for leakage.....gal.
(*e*) Total weight of water discharged, as measured.....lb.
(*f*) Total weight of water discharged, by calculation from plunger displacement, corrected for leakage.....lb.

Hourly Quantities

(23) Total water fed to boilers or drawn from surface condenser per hour.....lb.
(24) Total dry steam consumed for all purposes per hour (Item 21 ÷ Item 8) ....lb.
(25) Steam consumed per hour for all purposes foreign to main engine.....lb.
(26) Dry steam consumed by engine or turbine per hour (Item 24 − Item 25)....lb.
(*a*) Circulating water supplied to condenser per hour.....lb.
(27) Weight of water discharged per hour, by measurement.....lb.
(*a*) Weight of water discharged per hour, calculated from plunger displacement, corrected.....lb.

Hourly Heat Data

28) Heat units consumed by engine or turbine per hour [Item 26 × (total heat of 1 lb. of steam at pressure of Item 9, less heat in 1 lb. of water at temperature of Item 14)].....B.t.u.

Indicator Diagrams

(29) Mean effective pressure, each steam cylinder.....lb.
(*a*) Mean effective pressure, each water cylinder, if any.....lb.

Speed and Stroke

(30) Revolutions per minute.....r.p.m
(*a*) Number of single strokes per min.....(*b*) Average length of stroke .....ft

Power

(31) Indicated horsepower developed.....i.hp.
(*a*) Brake horsepower consumed by pump.....hp.
(32) Water horsepower[2].....hp.
(33) Friction horsepower (Item 31 − Item 32).....hp.
(34) Percentage of indicated horsepower lost in friction .....per cent

[1] Leakage of an inside plunger may best be determined by removing the cylinder head and bolting a wide board over the lower part of cylinder end to form a dam, in which an overflow pipe is inserted. The plunger is then blocked (preferably at some intermediate point of the stroke), and water admitted behind it from the force main, at full pressure. The leakage is caught from the overflow pipe in barrels and measured. If possible, tests should be made with the plunger in various positions. If the cylinder head is difficult to remove, the leakage may be measured through one of the openings provided for inspecting the suction valves. Any leakage of valves should be remedied before making the plunger test. Leakage of discharge valves will be shown by water passing down into the empty cylinder at either end when they are under pressure; and of suction valves by the disappearance of water which covers them. If valve leakage is found which cannot be remedied, the water thus lost should also be measured. One method consists in measuring the amount of water required to maintain a certain pressure in the pump cylinder when this is introduced through a pipe temporarily erected, no water being allowed to enter through the discharge valves of the pump.

[2] Water hp. = ft.-lb. of work per min. ÷ 33,000. Work done per min., ft.-lb. = net area of plunger *A* (sq. in.) × total head *H*, lb. per sq. in. (= pressure on force main + pressure on suction main + pressure equivalent to the head or vertical distance between the centers of the two gages) × length of stroke *S*, ft. × number of single strokes per min., *N*; corrected for leakage.

CAPACITY[1]

(35) Number of gallons of water discharged in 24 hr., as measured.....gal.
(*a*) Number of gallons of water discharged in 24 hr., calculated from plunger displacement, corrected.....gal.
(*b*) Gallons of water discharged per min., as measured.....gal.
(*c*) Gallons of water discharged per minute, calculated from plunger displacement, corrected.....gal.

ECONOMY RESULTS

(36) Heat units consumed per indicated horsepower-hour.....B.t.u.
(37) Heat units consumed per water horsepower-hour.....B.t.u.
(*a*) Dry steam consumed per indicated horsepower-hour.....lb.
(*b*) Dry steam consumed per water horsepower-hour.....lb.

EFFICIENCY RESULTS

(38) Thermal efficiency referred to indicated horsepower [(2546.5 ÷ Item 36) × 100].....per cent
(*a*) Thermal efficiency referred to water horsepower [(2,546.5 ÷ Item 37) × 100].....per cent
(*b*) Mechanical efficiency [(100 × Item 32)/Item 31].....per cent
(*c*) Pump efficiency [(100 × Item 32)/Item 31*a*].....per cent

DUTY

(39) Duty per 1,000,000 heat units.....ft.-lb.

WORK DONE PER HEAT UNIT

(40) Foot-pound of work per B.t.u. (1,980,000 ÷ Item 37).....ft.-lb.

SAMPLE DIAGRAMS

(41) Sample indicator diagrams from each steam and pump cylinder.....

NOTE.—The items relating to indicator diagrams and indicated horsepower are to be used only in the case of reciprocating machines. The indicated horsepower on which the economy results are based is that of the main engine, given by Item 31.

Operating Direct-acting Pumps.—Small pumps are provided with a pet cock on the discharge chamber for blowing out the air. On larger pumps a starting valve, or waste valve, should be placed on the discharge pipe inside of the gate and check valves. Prime and start the pump with the waste valve open to work the air out of the pump chambers. If there is a check valve, this will open automatically when the right speed is reached, but if there is only a gate valve, a gage should be placed at some point inside of the gate valve, which will indicate the proper time to open the gate valve, *viz.*, when the pressures on both sides are balanced. Both sides of a duplex pump should be packed alike, or one side will travel at a higher velocity than the other.

Before opening the throttle valve blow out the steam pipe. If cylinders are jacketed admit steam to the jackets some time before starting. If air pump is independent start same and get up a vacuum before starting the main engine. After stopping open all blowoffs and drain valves. If it is to be idle for some time open the cock of the oil cup, so as to let the oil flow into the steam chest, then let the pump make a few strokes to distribute the oil well over the inside of the steam end. When a pump is to be drowned, flood cylinders with oil and shut off steam from the surface.

[1] Capacity in gal. per 24 hr. = $74.8ASN$, corrected for leakage; in direct-connected engines, S = average length of stroke.

Keep the stuffing boxes well packed with a good quality of packing. Screw up the gland no tighter than necessary to stop leakage, and renew the packing before it becomes hard. Watch the rods and the outside-packed plungers to detect cutting. If pump is provided with a surface condenser, never exhaust into an empty condenser, as this will destroy the tube packing.

Operating Centrifugal Pumps.—Centrifugal pumps are primed before starting, either by means of an ejector with the discharge closed, or if a foot valve is provided at the end of the suction pipe, it may be filled with water through a priming pipe, the air escaping through a vent at the highest point of the casing. Start and bring up to speed, then open discharge valve. Always close the discharge valve before starting and stopping.

The stuffing boxes should be packed with square graphite packing. Round packing will wear grooves in the shaft. Obtain from the manufacturer a set of characteristic curves and compare with actual performance. If the readings do not agree with the curves examine pump for defects, such as stones in the impeller or broken blades. These curves will also assist in running the pump near the point of maximum efficiency.

Setting up Pumps.—The suction pipe should be large and as short and direct as possible. Bends should be of long radius. Lay to a uniform grade, rising towards the pump, so as to avoid air pockets. In connecting the steam pipe make due allowance for expansion. Place a throttle valve in the pipe close to the pump and provide means for draining the pipe with the valve closed. Before connecting the pipe to the steam chest blow out the steam thoroughly to keep dirt from getting into the cylinder.

The nozzles of a centrifugal pump as furnished by the manufacturer are necessarily computed for fairly high velocities. Increasers should be attached and the pipes should be made large enough to give velocities of from 4 to 5 ft. per second in the discharge pipe and from 3 to 4 ft. per second in the suction pipe. A strainer of liberal proportions should be at the bottom of the suction pipe, and the free opening through the strainer should be from 2 to 4 times that of the pipe. The openings through the strainer should be smaller than the smallest passage in the impeller. Install a gate valve and a check valve near the discharge nozzle with suitable drains, priming pipes and a pressure gage.

SECTION IV

THE TRANSPORTATION OF GASES

By Donald M. Liddell[1]

The transportation of gases may be roughly divided into the problems involving the use of air piped under more or less pressure, those of the removal of waste industrial gases by chimneys, and the carriage of highly-compressed gases in tanks.[2]

For low pressures coming under the first case, fans will ordinarily be used for pressures up to 16 oz. They are particularly used for ventilation, for moving shavings or other light material, and for supplying the blast in small foundry cupolas. Jet blowers may also be used for low pressures. Rotary blowers are ordinarily used for pressures from 1 to 10 lb., and piston compressors are used for still higher pressures and where positive action must be secured. Centrifugal compressors (turbo-blowers) are now also used for high-pressure work—an innovation of the last few years.

Table 1.—Weight of Air at Various Temperatures, Pressures, and Degrees of Humidity

Temperature, degrees Fahrenheit	Weight of 1 cu. ft. of dry air (at 14 lb. per square inch or 28.5 in. of Hg.), pounds	Increase or decrease of weight for each 0.1 lb. change in pressure, pounds	Increase or decrease of weight for each 1 in. of Hg. change of pressure, pounds	Decrease of weight for each 10 per cent increase in relative humidity, pounds
32	0.07688	0.000549	0.002698	0.000019
35	0.07642	0.000546	0.002681	0.000021
40	0.07565	0.000540	0.002654	0.000025
45	0.07490	0.000535	0.002628	0.000030
50	0.07417	0.000530	0.002602	0.000035
55	0.07340	0.000525	0.002580	0.000040
60	0.07272	0.000520	0.002554	0.000051
65	0.07203	0.000515	0.002530	0.000059
70	0.07134	0.000510	0.002506	0.000070
75	0.07068	0.000505	0.002482	0.000081
80	0.07003	0.000500	0.002457	0.000095
85	0.06938	0.000495	0.002432	0.000111
90	0.06875	0.000490	0.002408	0.000127
95	0.06811	0.000485	0.002384	0.000147
100	0.06752	0.000480	0.002359	0.000172
105	0.06694	0.000475	0.002334	0.000199

[1] Weld & Liddell, Engineers and Economists, 2 Rector St., N. Y.

[2] Acknowledgment is here made to Marks' "Mechanical Engineers' Handbook" and to the "Standard Handbook for Electrical Engineers" for much of the material in this chapter. The individuals whose work was thus used are: H. J. Thorkelson, L. C. Loewenstein, E. E. Williams, Joseph H. Brown, Jr., and R. J. S. Pigott.—Editor.

General Properties of Gases.—Gases when compressed or expanded isothermally approximately follow the law $pv = RmT$, where p = pressure, v = volume, R is the so-called gas constant (approximately 0.08207), m the mass of the gas in grams and T the absolute temperature (Boyle's law). A more nearly correct expression is given by van der Waals' equation $(p + \frac{a}{v^2})(v + b) = k$, where a and

TABLE 2.—SPECIFIC GRAVITY AND ABSOLUTE WEIGHT OF GASES

Gas	Formula	Molecular weight O = 16	Weight of 1 l. in grams at 0°C. and 760 mm. pressure	Specific gravity Air = 1	Weight of 1 cu. ft. in pound at 32°F. and 29.92 in. pressure	Critical temperature, degree Centigrade	Critical pressures atmospheres
Acetylene	C_2H_2	26.016	1.1708	0.90561	0.07309	35.5	61.7
Air	.....	.....	1.2928	1.0000	0.08071	−220.0	39.0
Aldehyde	C_2H_4O	44.032	1.9811	1.5324	0.12368		
Ammonia	NH_3	17.034	0.7708	0.59623	0.04812	131.0	113.0
Alcohol, ethyl	C_2H_5OH	46.048	2.0862	1.6137	0.13024	243.1	62.96
Alcohol, amyl	$C_5H_{11}OH$	88.096	4.0696	3.1479	0.25406		
Alcohol, methyl	CH_3OH	32.032	1.4483	1.1203	0.09042		
Argon	Ar	39.88	1.7809	1.3776	0.11118	−122.4	48.0
Arsine	AsH_3	77.984	3.4589	2.6755	0.21593		
Benzene	C_6H_6	78.048	3.5821	2.7708	0.22362	288.5	47.9
Boron chloride	BCl_3	117.38	5.09	3.937	0.1377		
Boron fluoride	BF_3	68.00	2.99	2.312	0.1867		
Bromine	Br_2	159.84	7.1437	5.5258	0.44597	302.2	
Butane	C_4H_{10}	58.08	2.65	2.050	0.1654		
Carbon dioxide	CO_2	44.00	1.9768	1.5291	0.12341	31.1	73.0
Carbon monoxide	CO	28.00	1.2504	0.96720	0.07806	−133.5	35.5
Carbonyl chloride	$COCl_2$	98.92	4.47	3.457	0.2791		
Carbonyl sulphide	COS	60.07	2.721	2.1047	0.16987	105.0	
Cyanogen	C_2N_2	52.05	2.335	1.806	0.14577		
Chlorine	Cl_2	70.92	3.222	2.4923	0.20114	146.0	83.9
Chlorine monoxide	Cl_2O	86.92	3.8820	3.0028	0.24235		
Chlorine dioxide	ClO_2	67.96	3.0192	2.3354	0.18848		
Ethane	C_2H_6	30.048	1.3562	1.0496	0.08467	32.1	49.0
Ethylene	C_2H_4	28.032	1.2609	0.97532	0.07872	9.5	50.8
Fluorine	F_2	38.00	1.635	1.2647	0.1021		
Helium	He	4.002	0.1782	0.1378	0.01112	−267.8	2.26
Hydrobromic acid	HBr	80.928	3.50	2.707	0.2185		
Hydrochloric acid	HCl	36.468	1.6392	1.26794	0.10233	51.8	83.6
Hydrofluoric acid	HF	20.008	0.9220	0.71318	0.05756		
Hydriodic acid	HI	127.928	3.657	2.8287	0.22830	150.7	
Hydrogen	H_2	2.016	0.08987	0.069516	0.005610	−241.1	11.0
Hydrogen arsenide	AsH_3	77.984	3.4589	2.6755	0.21593		
Hydrogen selenide	H_2Se	81.216	3.628	2.80639	0.22650	137.0	91.0
Hydrogen sulphide	H_2S	34.086	1.539	1.1904	0.09607	100.4	89.3
Hydrogen phosphide	PH_3	34.064	1.5293	1.18293	0.09547		
Hydrogen telluride	H_2Te	129.516	5.80	4.486	0.3621		
Hydrocyanic acid	HCN	27.018	1.226	0.9483	0.05920		
Iodine	I_2	253.84	11.271	8.7183	0.70363	512.0	
Krypton	Kr	82.92	3.708	2.8682	0.23148	− 62.5	41.24
Methane	CH_4	16.032	0.7168	0.55446	0.04475	− 81.8	54.9
Methyl chloride	CH_3Cl	50.484	2.3045	1.78261	0.14387		
Mercury	Hg	200.6	9.0210	6.97850	0.56317	1270.0	
Neon	Ne	20.0	0.9002	0.69634	0.05620	−205.0	29.0
Nitrogen	N_2	28.02	1.2057	0.93265	0.07527	−145.1	33.6
Nitrous oxide	N_2O	44.02	1.9782	1.53021	0.12350	36.5	71.95
Nitric oxide	NO	30.01	1.3402	1.03669	0.08367	− 33.5	71.2
Nitrogen tetroxide	N_2O_4	92.02	4.1133	3.18178	0.25679		
Nitrogen tetroxide	NO_2	46.01	2.0567	1.59092	0.12840		
Nitrosyl chloride	NOCl	65.47	2.9253	2.26282	0.18262	167.0	
Oxygen	O_2	32.00	1.4291	1.02803	0.08921	−118.8	50.8
Pentane	C_5H_{12}	72.096	.....	.....	.....	386.0	34.0
Phosphine	PH_3	34.064	1.5193	1.09788	0.09487	51.3	64.5
Phosphorus	P_4	124.16	5.6318	4.35639	0.35158		
Propane	C_3H_8	44.064	1.9660	1.558	0.12273		
Propylene	C_3H_6	42.048	1.8783	1.45293	0.11726		
Silicon fluoride	SiF_4	104.3	4.684	3.06490	0.29093		
Sulphur dioxide	SO_2	64.07	2.9266	2.26390	0.18264	157.0	78.0
Xenon	Xe	130.2	5.851	3.7524	0.36527	14.7	43.5
Radium emanation	Nt	222.4	9.727	7.5421	0.60724		
Water	H_2O	18.016	0.8063	0.6237	0.050336	364.3	194.6

b are constants varying with different gases (see p. 190 for a table of these constants). If a gas be expanded or compressed adiabatically (without gain or loss of heat from or to external objects), the equation is $px^{\gamma} = RmT$, where $\gamma = 1.406$ approximately.

Accompanying tables give the weight of a cubic foot of air at various temperatures and pressures (Table 1), the specific gravity of various other gases (Table 2), and the weight of saturated air per cubic foot (Table 3), together with other useful data concerning gases. Gas measurements are reduced for comparison to their calculated volume at 0°C. and 760 mm. Hg. pressure (29.92 in.) which are known as normal temperature and pressure (N. P. T.)

The critical temperature of a gas is such a temperature that above it, no pressure, however great, will liquefy the gas. The critical pressure is that pressure at which gas at the critical temperature begins to liquefy. (See Table 2, also p. 657.)

TABLE 3.—WEIGHT OF SATURATED AIR IN POUNDS PER CUBIC FOOT AT DIFFERENT BAROMETRIC PRESSURES

Temperature, degrees Fahrenheit	Barometer readings, inches of mercury								
	28.5	29.0	29.5	29.7	29.9	30.1	30.3	30.5	31.0
30	0.07703	0.07839	0.07974	0.08028	0.08083	0.08137	0.08191	0.08245	0.08381
35	0.07621	0.07756	0.07890	0.07943	0.07997	0.08051	0.08104	0.08158	0.08292
40	0.07541	0.07674	0.07806	0.07859	0.07913	0.07966	0.08019	0.08072	0.08205
45	0.07461	0.07592	0.07724	0.07776	0.07829	0.07881	0.07934	0.07986	0.08118
50	0.07381	0.07512	0.07642	0.07694	0.07746	0.07798	0.07850	0.07902	0.08032
55	0.07302	0.07431	0.07560	0.07612	0.07663	0.07715	0.07766	0.07818	0.07947
60	0.07224	0.07352	0.07479	0.07530	0.07581	0.07632	0.07683	0.07734	0.07862
65	0.07145	0.07272	0.07398	0.07449	0.07499	0.07550	0.07600	0.07651	0.07777
70	0.07067	0.07192	0.07317	0.07367	0.07417	0.07467	0.07518	0.07568	0.07693
75	0.06988	0.07112	0.07236	0.07286	0.07335	0.07385	0.07434	0.07484	0.07608
80	0.06909	0.07032	0.07155	0.07204	0.07253	0.07302	0.07351	0.07400	0.07523
85	0.06829	0.06950	0.07072	0.07121	0.07170	0.07218	0.07267	0.07316	0.07437
90	0.06748	0.06868	0.06989	0.07037	0.07085	0.07133	0.07182	0.07230	0.07351
95	0.06665	0.06785	0.06904	0.06592	0.07000	0.07048	0.07095	0.07143	0.07263
100	0.06581	0.06700	0.06818	0.06866	0.06913	0.06960	0.07008	0.07055	0.07174

Specific Heat.—The instantaneous specific heat of dry air is given by F. G. Swann as $0.24112 + 0.000009\,t$, and the specific heat of water vapor as $0.4423 + 0.00018\,t$, where t is the temperature in degrees Fahrenheit. The mean specific heat of air with any degree of saturation may then be found by multiplying the

TABLE 4.—SPECIFIC HEATS OF DRY AND SATURATED AIR (CONST. PRESS.)

Temperature, degrees Fahrenheit	Specific heat of		Temperature, degrees Fahrenheit	Specific heat of	
	Dry air	Saturated air		Dry air	Saturated air
60	0.2417	0.244	85	0.2419	0.2474
65	0.2417	0.2447	90	0.2419	0.2486
70	0.2417	0.2452	95	0.2420	0.2498
75	0.2418	0.2458	100	0.2420	0.2512
80	0.2418	0.2466			

weight of air by its specific heat and adding to this the product of the weight of water vapor and its specific heat and dividing the sum by the weight of the mixture. Table 4 is calculated on that basis.

The mean specific heat of air having any relative humidity may be found from Table 4 by interpolation. *E.g.*, for air of 50 per cent relative humidity and at 80°F. the mean specific heat will be $0.2418 + 0.40 \times (0.2466 - 0.2418)$, or 0.2437.

The variation of the specific heat of air with pressure has been investigated by Holborn and Jakob (*Z. V. D. I.*, Vol. **58**, p. 1436). The mean specific heat at constant pressure for the temperature range 20–100°C. (68–212°F.) is given by the equation $10^4 c_p = 2{,}413 + 286p + 0.0005\ p^2 - 0.00001\ p^3$, where p is the pressure in kilograms per square centimeter. The experimental values are given below:

Pressure, pounds per square inch absolute	14.2	356	711	1,422	2,133	2,844
c^p	0.2415	0.2490	0.2554	0.2690	0.2821	0.2925

The mean specific heats of various gases are given in Table 5.

Table 5.—Mean Specific Heats of Gases

	Under constant pressure	Under constant volume	γ
Acetylene	.....	.....	1.26
Air, 20°C.	0.2417	01724.	1.402
Ammonia	0.5356	0.391	1.336
Argon, 20 to 90°C.	0.123	.....	1.66
Benzene, 34 to 115°	0.299	.....	(20°) 1.40
Bromine, 19 to 388°	0.0555	0.0429	.....
Carbon dioxide, 0°	0.2010	0.172	1.30
Carbon disulphide, 86 to 190°	0.1596	0.131	1.239
Carbon monoxide, 23 to 99°	0.2425	0.1736	1.401
Chlorine	0.1241	0.0928	1.33
Ethane	.....	.....	1.22
Ethylene	0.404	.....	1.264
Hydrogen	3.4090	2.411	1.42
Iodine, 206 to 377°C.	0.034	.....	.....
Methane	0.5929	0.486	1.313
Nitrogen, 0°C.	0.2350	0.1727	1.41
Nitrous oxide	0.2262	0.181	1.324
Oxygen	0.2175	0.1723	1.41–
Sulphur dioxide	0.1544	0.123	(500°) 1.2
Water	0.4805	0.370	1.305
Hydrochloric acid	0.1867	.....	1.40
Nitric oxide, 13 to 172°	0.232	.....	1.394
Nitrogen peroxide, 27 to 67°	1.625	.....	(150°) 1.31
Sulphuretted hydrogen, 20 to 206°	0.245	.....	1.340
Turpentine, 179 to 249°	0.506	.....	.....

Calculations of the work done by the expansion of air are greatly facilitated by the diagram, Fig. 1, adapted from the "Entropy-Log. Temperature Diagram for Air" by Prof. C. R. Richards (*Bulletin* No. 63, University of Illinois Experiment Station).

In Fig. 1 the vertical lines represent the volume in cubic feet occupied by 1 lb.

of air. The lines slightly inclined to the vertical represent the absolute pressure in pounds per square inch. The lines at an angle of 45 deg. represent temperature in degrees Fahrenheit and intrinsic (internal) energy in B.t.u., and the dotted inclined lines represent entropy. These last lines are paralleled in finding the results of an adiabatic change, while if the change follows a $p_1V_1^n = p_2V_2^n$ path, the effects are studied by paralleling lines indicated at the margin for various values of n. For example, an inspection of Fig. 1 shows that 1 lb. weight of air at a pressure of 125 lb. absolute per square inch and occupying a volume of 1.5 cu. ft. will have a temperature of 50°F. and contain 85.86 B.t.u. or 66,800 ft.-lb. of internal energy.

Furthermore, the adiabatic expansion of 1 lb. of air from a pressure of 125 lb. absolute per square inch and a temperature of 70°F. to a pressure of 14.7 lb. absolute

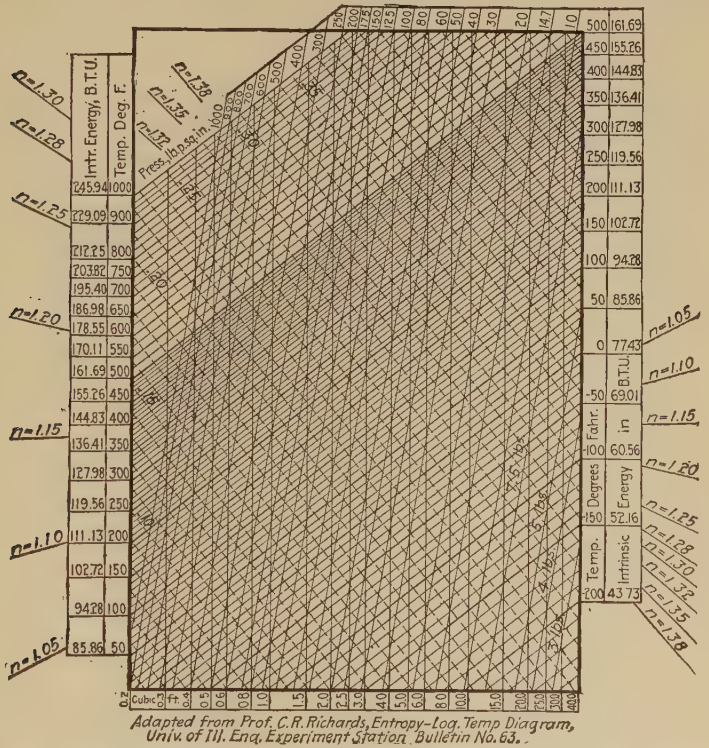


FIG. 1.—Diagram for determining the amount of energy stored in compressed air.

will result in a final temperature of −170°F. and the work done during the expansion would be (89.22 − 48.79 =) 40.43 B.t.u. or 31,454 ft.-lb. per pound of air. This expansion follows the dotted line of the diagram, but if the expansion followed the equation $p_1V_1^{1.2} = p_2V_2^{1.2}$, the available intrinsic energy per pound for any pressure range would be found by following a line parallel to the marginal line of the chart marked $n = 1.20$ from the upper to the lower pressure. *E.g.*, if such an expansion occurred between 125 lb. absolute per square inch and 70°F. and 14.7 lb. absolute, the resulting temperature would be −87°F. and the work done would be (89.22 − 62.76 =) 27.46 B.t.u. or 21,346 ft.-lb. per pound of air.

With adiabatic compression, the final temperature T_2 (degrees Fanrenheit absolute) is given by the formula $T_2 = T_1(p_2/p_1)^{0.29}$ for single-stage compression; by

$T_2 = T_1\sqrt{(p_2/p_1)^{0.29}}$ for two-stage compression, and by $T_2 = T_1\sqrt[3]{(p_2/p_1)^{0.29}}$ for three-stage. Values of T_2 may be obtained readily from the chart devised by F. W. O'Neill and shown in 2, in which the ordinates give the factors by which the initial absolute temperature T_1 (degree Fahrenheit) is to be multiplied to obtain the final absolute temperature T_2 for various ratios of p_2 to p_1.

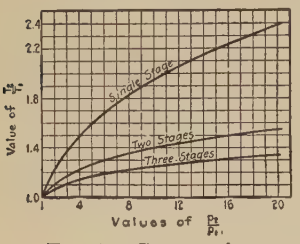


FIG. 2.—Compression temperatures.

Fans.—As said above, these are used only at low pressures. They are of two types, the propeller and the centrifugal. The former moves the air in a direction parallel to its axis of rotation, the effect being similar to that of a ship's propeller. The centrifugal fans have an inlet at the center and discharge at the periphery.

Fans may operate with equal efficiency at any speed which their mechanical strength safely allows. They may also discharge, in practice, through orifices varying in area from that of their full discharge, to a very small per cent of the full outlet. The efficiency varies with different orifice areas, but not according to any fixed law for all designs.

The horsepower for any fan or blower, if the area of the discharge orifice remains unchanged, varies directly as the cube of the speed; also, if the area of the discharge orifice remains unchanged and the speed is constant, the horsepower varies directly with the density of the gas being handled.

The horsepower of a centrifugal fan, if the speed remains unchanged, generally decreases as the area of discharge orifice is decreased. In some designs the horsepower with rated orifice is not over one-third of the horsepower with orifice equal to area of fan outlet, and in other designs it is very little less at rated orifice than with unrestricted orifice.

The horsepower of a propeller fan, if the speed remains unchanged, increases as the area of the discharge orifice is decreased. In some designs the horsepower with rated orifice is not over one-quarter of the horsepower with complete restriction.

The propeller fan is used in heating, ventilating, cooling, drying, etc., rarely for pressures exceeding 1 in. of water Their rated capacities are given in the accompanying table.

TABLE 6.—RATED CAPACITIES OF PROPELLER FANS

Wheel diameter, feet	Free delivery			Static pressure, inches								
				0.25			0.5			1.0		
	Cubic feet per minute	Revolutions per minute	Brake horse power	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower
2	4,000	170	0.24	2,100	710	0.53	3,000	1,000	1.5	4,250	1,420	4.2
3	9,000	465	0.54	4,800	470	1.2	6,750	670	3.4	9,500	940	9.6
4	16,000	350	0.95	8,500	355	2.1	12,000	500	6.0	17,000	710	17.0
6	36,000	233	2.15	19,000	235	5.1	27,000	335	14.5	38,000	470	41.0
8	64,000	175	3.8	34,000	175	8.5	48,000	250	24.0	68,000	355	68.0

Approximate Horsepower of Propeller Fans.—The horsepower required to drive any propeller fan may be represented by an expression of the form,

$$\text{horsepower} = K_1 \left(\frac{T}{1000}\right)^3 A \left(\frac{w}{0.075}\right)$$

Where K_1 is a constant depending upon the design and upon other conditions, T is the peripheral velocity or the tip speed of the wheel in feet per minute; A is the gross area of the wheel in square feet; w is the absolute density of the gas handled in pound per cubic feet; 0.075 is the weight of standard air at 65°F. in pounds per cubic foot.

TABLE 7.—CONSTANTS FOR PROPELLER FANS
(B. F. Sturtevant Company)

Type of fan	K_1
Sturtevant propeller	0.0006
Multiblade disc	0.0012
Blackman propeller	0.0012
Davidson propeller	0.0015

Centrifugal fans find a much wider application than do the propeller. They are used up to pressures of about 16 oz. or about 27 in. of water.

TABLE 8.—TABLE OF CONSTANTS FOR CENTRIFUGAL FANS
(B. F. Sturtevant Company)

Type of fan	K
Paddle-wheel with 6 to 10 straight blades and $W = 0.45D$	0.0060
Steel pressure with 24 backward curved blades and $W = 0.15D$	0.0050
Slow speed conveying with 18 forward curved blades and $W = 0.25D$	0.0100
Single-inlet multiblade-drum with 60 narrow curved blades, inclined forward and $W = 0.50D$	0.0350
Double-inlet multiblade-drum with 60 narrow curved blades, inclined forward and $W = 0.25D$	0.0250
Double-inlet multiblade-drum with 60 narrow curved blades, inclined forward and $W = 0.50D$	0.0400
Double-inlet multiblade-drum with 60 narrow curved blades, inclined forward and $W = 1.00D$	0.0350

TABLE 9.—CONVERSION TABLES FOR AIR PRESSURES

PRESSURES IN INCHES OF WATER CORRESPONDING TO OUNCES PER SQUARE INCH

Pressure, ounces per square inch	Decimal parts of an ounce									
	0.0	0.1	0.2	0.3	0.4	0.5	0.6	0.7	0.8	0.9
0	.....	0.17	0.35	0.52	0.69	0.87	1.04	1.21	1.38	1.56
1	1.73	1.90	2.08	2.25	2.42	2.60	2.77	2.94	3.11	3.29
2	3.46	3.63	3.81	3.98	4.15	4.33	4.50	4.67	4.84	5.01
3	5.19	5.36	5.54	5.71	5.88	6.06	6.23	6.40	6.57	6.75
4	6.92	7.09	7.27	7.44	7.61	7.79	7.96	8.13	8.30	8.48
5	8.65	8.82	9.00	9.17	9.34	9.52	9.69	9.86	10.03	10.21
6	10.38	10.55	10.73	10.90	11.07	11.26	11.43	11.60	11.77	11.95
7	12.11	12.28	12.46	12.63	12.80	12.97	13.15	13.32	13.49	13.67
8	13.84	14.01	14.19	14.36	14.53	14.71	14.88	15.05	15.22	15.40
9	15.57	15.74	15.92	16.09	16.26	16.45	16.62	16.79	16.96	17.14

PRESSURE IN OUNCES PER SQUARE INCH CORRESPONDING TO INCHES OF WATER

Head, inches	Decimal parts of an inch									
	0.0	0.1	0.2	0.3	0.4	0.5	0.6	0.7	0.8	0.9
0	.....	0.06	0.12	0.17	0.23	0.29	0.35	0.40	0.46	0.52
1	0.58	0.63	0.69	0.75	0.81	0.87	0.93	0.98	1.04	1.09
2	1.16	1.21	1.27	1.33	1.39	1.44	1.50	1.56	1.62	1.67
3	1.73	1.79	1.85	1.91	1.96	2.02	2.08	2.14	2.19	2.25
4	2.31	2.37	2.42	2.48	2.54	2.60	2.66	2.72	2.77	2.83
5	2.89	2.94	3.00	3.06	3.12	3.18	3.24	3.29	3.35	3.41
6	3.47	3.52	3.58	3.64	3.70	3.75	3.81	3.87	3.92	3.98
7	4.04	4.10	4.16	4.22	4.28	4.33	4.39	4.45	4.50	4.56
8	4.62	4.67	4.73	4.79	4.85	4.91	4.97	5.03	5.08	5.14
9	5.20	5.26	5.31	5.37	5.42	5.48	5.54	5.60	5.66	5.72

FUNDAMENTAL FORMULAS

Notation.

H = head of air, feet
h = velocity head, inches of water
h_s = static head, inches of water
h_i = impact head, inches of water
v = velocity, feet per second
V = velocity, feet per minute
W = weight of air, pound per cubic foot, as handled
P = pressure, pound per square foot
A = area of pipe, square feet
a = gross area of fan wheel, square feet
Q = volume, cubic feet per minute
q = volume, cubic feet per second
t = temperature, degrees Fahrenheit
B = Barometer reading, inches of mercury
a.hp. = air horsepower
b.hp. = brake horsepower
E = mechanical efficiency
N = revolutions per minute
D = diameter of fan wheel, feet
r = radius of fan wheel, feet
b = width of fan wheel, feet
M = manometric efficiency
C_1 = tip speed, feet per second
T = tip speed, feet per minute

Velocity.—From the formula $v = \sqrt{2gH}$ the velocity of air at standard (70°F., 29.92 in.) conditions ($W = 0.075$) is given by $V = 4000\sqrt{h}$. For any given air temperature t and pressure B, neglecting the slight effect of a variable humidity, $V = 956\sqrt{(460 + t)h/B}$.

Pressure is generally expressed in inches of water or in ounces per square inch. 1 oz. per square inch = 1.732 in. of water $H = 5.2h/W$; $P = HW$, and $P = 5.2h$. For air at density $W = 0.075$, a head of 1 in. of water equals that of 69.3 ft. of air. Three pressures must be considered in a column of moving air, namely, static, velocity and impact. The first represents the compression, the second kinetic energy of the blast, and the third the total pressure or the sum of the static and velocity pressures. The static pressure in a system through which air passes is often referred to as "maintained resistance."

Air Horsepower.—The horsepower in a column of moving air is a hp. $= V\,AP/33{,}000 = QP/33{,}000 = 5.2Qh_i/33{,}000 = 0.1728Ah_i\sqrt{h/W}$. The horsepower required to drive a centrifugal fan may be roughly figured as $K\left(\frac{T}{1{,}000}\right)^3 a\left(\frac{w}{0.075}\right)$.

Mechanical Efficiency is the ratio of air horsepower to brake horsepower. Sometimes the air horsepower is computed by use of only the static pressure—positive, negative or both—against which the fan operates; it being assumed that static pressure represents the useful pressure and that the kinetic energy or velocity pressure is thrown away on leaving the system and entering the atmosphere. Since in most fan systems the velocity at the point of delivery into the atmosphere is considerably lower than at the fan outlet, the kinetic energy or velocity pressure which can rightfully be deducted from the impact or total head produced by the fan is only that existing at the point of leaving the system, and not the velocity pressure at the fan outlet. The impact or total head produced by the fan is more generally used in computing the air horsepower.

TABLE 10.—VELOCITY OF AIR DUE TO PRESSURE
(Air at 65°F.; barometer reading, 29.92 in.)

Pressure, inches of water	Velocity, feet per minute	Pressure, inches of water	Velocity, feet per minute	Pressure, inches of water	Velocity, feet per minute	Pressure, inches of water	Velocity, feet per minute
0.1	1,265	0.9	3,790	1.7	5,210	3.25	7,210
0.2	1,790	1.0	4,000	1.8	5,360	3.50	7,490
0.3	2,190	1.1	4,190	1.9	5,510	3.75	7,750
0.4	2,530	1.2	4,380	2.00	5,650	4.00	8,000
0.5	2,830	1.3	4,560	2.25	6,000	4.25	8.250
0.6	3,100	1.4	4,730	2.50	6,320	4.50	8,490
0.7	3,350	1.5	4,900	2.75	6,630	4.75	8,720
0.8	3,580	1.6	5,050	3.00	6,930	5.00	8,950

Manometric Efficiency or pressure efficiency is the pressure developed by the fan divided by the pressure against a plane surface due to a velocity equal to the peripheral speed of the fan wheel. $M = gH/C_1^2$. Manometric efficiency is defined by some engineers as $2gH/C_1^2$.

Volumetric efficiency is not really an efficiency and might better be called volumetric capacity. It is defined as the quantity of air delivered per revolution divided by the overall cubical contents of the wheel, and is represented by the formula Vol. Eff. $= Q/\pi r^2 bN$.

Performance of Fans at Various Air Temperatures.—In Table 11 are given factors for determining the influence of the air temperature on the volume of a

given weight (col. 2) and on the weight of a given volume (col. 3) of air; on the pressures (cols. 2 and 3) corresponding to constant weight and constant volume, respectively, of air handled by a fan; on the speeds (cols. 2 and 4) corresponding to constant weight and constant pressure, respectively, of air handled by a fan; on the power (cols. 3 and 5) corresponding to constant volume and constant weight, respectively, of air handled by a fan; and the power (col. 2) necessary to handle a given weight at a given pressure with a fan proportioned to operate at a given efficiency. All these factors are to be applied to the quantities corresponding to the standard air temperature of 65°F

TABLE 11.—FACTORS FOR DETERMINING THE PERFORMANCE OF FANS AT VARIOUS AIR TEMPERATURES (SEE TEXT)

Temperature, degrees Fahrenheit	2	3	4	5	Temperature, degrees Fahrenheit	2	3	4	5
30	0.94	1.07	0.97	0.87	325	1.50	0.67	1.22	2.24
40	0.96	1.05	0.98	0.91	350	1.55	0.65	1.24	2.38
50	0.97	1.03	0.99	0.95	375	1.59	0.63	1.26	2.54
60	0.99	1.01	0.99	0.98	400	1.63	0.61	1.28	2.69
65	1.00	1.00	1.00	1.00	425	1.68	0.60	1.30	2.85
70	1.01	0.99	1.01	1.02	450	1.73	0.58	1.32	3.02
80	1.03	0.97	1.02	1.06	475	1.78	0.57	1.33	3.18
90	1.05	0.95	1.03	1.11	500	1.83	0.54	1.35	3.38
100	1.07	0.93	1.04	1.15	525	1.88	0.53	1.37	3.52
125	1.12	0.89	1.06	1.25	550	1.93	0.52	1.39	3.72
150	1.17	0.86	1.08	1.36	575	1.98	0.50	1.41	3.90
175	1.21	0.83	1.10	1.47	600	2.02	0.49	1.42	4.10
200	1.26	0.80	1.12	1.58	650	2.12	0.47	1.46	4.50
225	1.30	0.77	1.14	1.71	700	2.21	0.45	1.49	4.90
250	1.35	0.74	1.16	1.83	750	2.31	0.43	1.52	5.32
275	1.40	0.71	1.18	1.96	800	2.41	0.41	1.55	5.78
300	1.45	0.69	1.20	2.10					

For the weight of 1 cu. ft. of air at any temperature, pressure and humidity within the usual range, see page 143.

Fan Characteristics.—The characteristics of any fan can best be shown graphically. The characteristic curves from any size of a series of similar fans suffice for the entire group. This follows from the fact that the area of the frictional surfaces varies directly with the area of the various passages through which air flows when passing through the fan. The different bases upon which these characteristic curves can be plotted are as stated in the following paragraphs.

Static No Delivery (S.N.D.).—This gives a useful basis for calculating the performance of various fans. Figure 3 is the chart of a fan known as the "steelplate" or "paddle-wheel" type. Data for these curves are obtained by running the fan at some convenient constant speed and taking a set of readings at each of six to eight air deliveries produced by as many different restrictions placed on the discharge and varying from a condition of closed outlet to wide open or free discharge. All data are taken at the fan outlet. The abscissas are the static pressures or maintained resis-

tances in percentages of the pressure which the fan will exert with its outlet completely closed; this latter pressure with a few exceptions is the maximum pressure a fan will exert at any given speed, and is symbolized S.N.D. Four curves are plotted, using as ordinates (1) the ratio of the velocity through the fan outlet to the peripheral speed of the wheel and designated V/T; (2) the mechanical efficiency E; (3) the ratio of

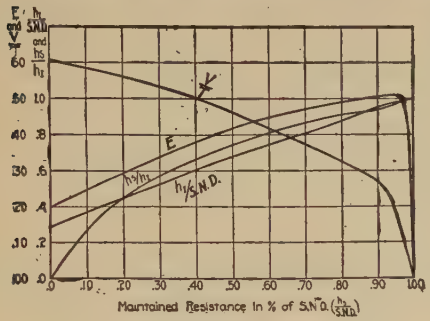


FIG. 3.—Characteristic curves of a steel-plate paddle-wheel fan.

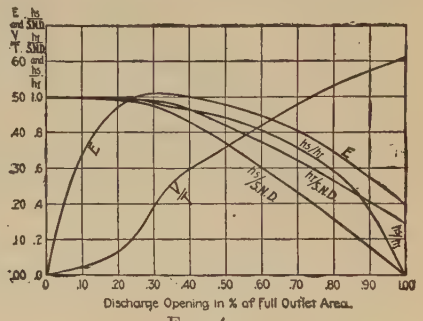


FIG. 4.

impact pressure to S.N.D., designated h_i/S.N.D.; and (4) the ratio of static pressure to impact pressure, designated h_s/h_i. From these curves can be computed the performance of any size of symmetrical fan under any conditions of pressure, volume or speed within its structural possibilities.

As an example of how these curves are applied, let it be assumed that a size of fan is required which will operate most efficiently when delivering 10,000 cu. ft. per minute against 2 in. static pressure. Assume the fan to operate at a mechanical efficiency of 50 per cent. At this efficiency the abscissa h_s/S.N.D. is 0.857, from which S.N.D. = 2.0/0.857 = 2.33 in. For the same efficiency h_i/S.N.D. = 0.923, therefore $h_i = 0.923 \times 2.33 = 2.15$ in.; $h_i = h_s + h$, hence $h = 0.15$ in.; and $V = 4000\sqrt{0.15} = 1{,}548$ ft. per minute. For 10,000 cu. ft. per minute the area of the fan outlet must then be 6.47 sq. ft. Knowing that for this particular design of fan the outlet area is expressed by the equation $A = 0.306D^2$, the diameter D is found to be 4.59 ft. Corresponding to h_s/S.N.D. = 0.857 is found a value of $V/T = 0.298$, from which the peripheral speed = 1,548/0.298 = 5,200 ft. per minute, and the speed of the fan is computed as 360 r.p.m. The other dimensions of the fan are found from the basis of design, in which these dimensions should be expressed as functions of the diameter of the wheel D. Air horsepower = a.hp. = $10{,}000 \times 2.15 \times 5.2/33{,}000 = 3.39$; b.hp. = 3.39/0.50 = 6.78.

As a second example suppose a fan with a wheel 3 ft. in diameter is to deliver 5,000 cu. ft. per minute against 2 in. maintained resistance or static pressure, leaving revolutions per minute and brake horsepower to be determined. From the equation $A = 0.306D^2$, $A = 2.76$ sq. ft., therefore $V = 1{,}810$ ft. per minute. From $V = 4{,}000\sqrt{h}$, $h = 0.205$ in. making $h_i = 2.205$ in. Here $h_s/h_i = 0.907$, at which point on the curves of Fig. 1 $V/T = 0.322$; hence $T = 5{,}620$ ft. per minute. As $D = 3$ ft., r.p.m. = 597. a.hp. = $5{,}000 \times 2.205 \times 5.2/33{,}000 = 1.74$. From the curves $E = 49.2$ per cent, hence b.hp. = 3.54.

The curves of Fig. 3 may also be plotted using for abscissas the static pressures or maintained resistances expressed in percentages of the pressure which would result from a velocity equal to the peripheral or tip speed of the blades and which is referred to as tip-speed pressure.

Discharge Opening in Percentage of Full Outlet.—Figure 4 is for the same fan as Fig. 3 and shows the same ordinates plotted against the discharge outlet area

(during the test) expressed as a percentage of the full outlet. As the majority of fan problems require solution on a basis of static pressure or maintained resistance rather than on a percentage of full outlet (a laboratory condition), Fig. 3 proves of greater utility. A curve of "discharge opening in percentage of full outlet" could be plotted on Fig. 3 as another ordinate against maintained resistance if desired.

Reduced Orifice.—This term is more common in European practice and is used as an abscissa against which are plotted volumetric, manometric and mechanical efficiencies. Reduced orifice is expressed by the equation $O_r = q/(r^2\sqrt{gH})$. Since this equation contains the factor r, it is evident that regardless of the size of fan of any given type the reduced orifice will be the same when the same relative restriction exists. This approaches the basis of Fig. 4, but instead of dealing with the percentage of opening of outlet, it deals with a value that is in reality some constant times the true percentage of full opening.

Equivalent Orifice.—This term is also more common in European practice and differs from "reduced orifice" in that the equation contains no term bearing on the proportions or size of the fan; hence the equivalent orifice is an actual quantity in square feet varying with the volume delivered. Equivalent orifice is expressed by the equation $O_e = q/(K\sqrt{2gH})$. It represents the area of a circular hole in a thin plate which will produce a restricting effect equal to that which exists in the system under consideration regardless of the nature of the actual restriction. The value of K generally used ranges from 0.60 to 0.65, making $O_e = q/0.65\sqrt{2gH}$. It is more of a laboratory characteristic than one adaptable to general use.

Volumetric and manometric efficiency curves could be plotted, but the usefulness of such curves is limited. Volumetric efficiency is a direct function of the V/T-curve and may be represented by the product of V/T and a constant, the latter depending on the design of the fan. As the volume delivered at a given revolution per minute varies as D^3, the volume per revolution $= K_1D^3V/T$, and since the overall cubical content of the wheel is K_2D^3, the volumetric efficiency $= KV/T$, where K equals K_1/K_2. K_1 and K_2 depend on the design and proportions of the fan. The volumetric efficiency is maximum with free intake and discharge and runs as high as 400 to 500 per cent in fans of the multiblade type designed for high pressures at low peripheral speeds, while it runs as low as 100 per cent in fans designed for low pressures at high peripheral speeds. The manometric efficiency is generally highest when the outlet is closed and lowest when maximum volume is delivered. There are, however, types where the maximum value occurs at light loads varying from ¼ to ½ maximum volumetric delivery. In cased fans of the multiblade type the maximum efficiency (gH/C_1^2) attained is about 150 per cent, while in open-running fans it is as low as 30 per cent. High mechanical efficiency (and generally high volumetric efficiency) is desirable, but with manometric efficiency it is usually the method of drive which determines the desirable value. The higher the speed of the driver the lower the manometric efficiency of the fan best adapted to the drive.

Effects of Variations in Revolutions Per Minute, Volume, Pressure, Horsepower, Efficiency, Etc.—Where a given fan is operating with a given restriction on its inlet or outlet, the volume delivered will vary directly with the revolutions per minute (N), the various pressures as N^2, and the air horsepower as N^3. The mechanical efficiency will remain the same, therefore the brake horsepower will also vary as N^3. The volume delivered by symmetrical fans will vary as D^2 for any given peripheral speed of the wheel and resistance or static pressure. Under these conditions the mechanical efficiency remains constant, since the area of friction surfaces varies directly with the change in volume. The horsepower will therefore vary as D^2. It is sometimes erroneously assumed that at very

high speeds a given fan will show a marked decrease in efficiency due to slippage, or what with ship propellers is known as cavitation. This, however, is not the case.

Methods of Testing Fans. *Air-tight-room Test.*—In this test the fan discharges into a closed room, the outlet of which is a sliding door or a suitable discharge pipe. The static pressure in the room represents the resistance against which the fan is operating. The sliding door is used to vary the output of the fan, but as the coefficient of efflux varies with different openings as a result of their varying shape, it is better to measure the volume delivered through a discharge pipe of such length as will practically eliminate eddies. The room should be of such size that there are no heavy swirls, and even though extra caution be used in making it tight, a leakage test should be made. With this method of testing it is impossible to get data for a free and unrestricted discharge, yet if the room is relatively large this condition is approached with reasonable closeness. Prof. R. J. Durley[1] lets the air discharge from a room or closed vessel through small circular orifices in thin plates of known coefficient of discharge. This, however, is essentially a laboratory method.

Discharge Pipe Test.—The discharge pipe, which is preferably of the shape and size of the fan outlet, should have a length of 15 to 20 diameters or 15 to 20 times the mean of the rectilinear dimensions. The Pitot tube should be placed at about 10 diameters from the fan outlet. Traverse readings are taken at this point with a Pitot tube. To get a correct average of the velocity pressure, square-root-of-mean-square-values should be used, but, even with a wide variation in the readings, the inaccuracy of using a plain arithmetical mean is not over 0.5 per cent. Center readings only can be taken, corrected by a coefficient for the pipe if such procedure seems desirable. Credit should be given the fan for the friction loss between the fan outlet and point of measurement. Galvanized iron is the common material of the discharge duct, and the friction loss in inches of water can be computed from the formula $H = 4flv^2/d$, where H = head in feet of air, f = coefficient of friction = 0.0001 (about) for galvanized iron pipe, l = length (ft.), v = velocity (feet per second), and d = diameter of discharge (ft.). This formula may be simplified and written as $h_f = lh/kd$, where k = a constant. h_f = head lost in friction, and h = velocity head in the pipe, both in inches of water. The value of k for smooth galvanized-iron piping ranges from 50 to 60, while for 12-in. swaged pipe it may be as low as 40 and in some cases is still lower. Readings taken at conical mouthpieces at the end of this discharge pipe are not as accurate as those taken with a Pitot tube in the pipe.

Suction Pipe Test.—A similar pipe can be placed on the inlet and Pitot tube readings taken therein. Here the true resistance against which the fan works will be the *impact reading.*

Fan Design.—The matter of fan design will be left to the mechanical engineer and only treated here so far as it will involve choice between different fans. The use of a double discharge, the discharges being 180° apart will decrease both the manometric and volumetric efficiencies.

The point where the scroll or spiral discontinues its approach to the circumference of the wheel is called the cutoff point. In some types of fan nearly half the diameter of the wheel is exposed when looking along the axis of discharge, while the other extreme is found where no part of the wheel is exposed, in which case a tangent to the wheel passing through the cutoff point is parallel to the axis of discharge. A fan with a large exposure of wheel will deliver, with very little or no restriction to the flow of air, a relatively greater volume at a given number of revolutions per minute than

[1] *Trans. A. S. M. E.*, Vol. **27.**

the fan in which there is little or no exposure of the wheel, while the latter will deliver a relatively greater volume against average restrictions or resistances met with in practice. The following table, from tests on a multiblade fan, shows the difference in efficiencies at various pressures or resistances at a given speed in revolutions per minute when 25 per cent of the diameter of wheel is exposed [col. (*a*)] and when a tangent to the wheel passing through the cutoff point is also parallel to the axis of discharge [col. (*b*)]. The same wheel and casing, except for the cutoff piece, were used in each case. Figures 5 and 6 show the feature being considered.

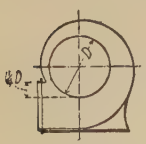


FIG. 5.

FIG. 6.

Spirals of fan casings.

Static pressure, inches of water	Efficiencies, per cent					
	Mechanical		Volumetric		Manometric	
	(*a*)	(*b*)	(*a*)	(*b*)	(*a*)	(*b*)
0.6	40.0	40.0	396	383	79.4	76.8
1.0	47.2	49.5	335	340	94.6	94.6
1.1	48.2	51.0	305	322	96.2	98.5
1.2	48.5	51.5	262	301	96.0	100.7
1.3	47.0	51.5	189	271	94.7	102.4
1.4	41.0	50.0	116	215	96.0	103.7

When a fan operates against considerable restriction or resistance to the flow of air, there is a strong tendency in the air to leak back through the wheel at the point where it is about to leave the spiral to pass through the fan discharge. If the delivery is restricted, it leaks back through the wheel at the point of maximum pressure. The above table shows how the efficiencies drop off in the fan of Fig. 5 as compared with that of Fig. 6. The maximum pressure in the air is at the fan discharge, while from the bottom of the wheel to the cutoff point (that is, the exposed portion of the wheel) the air has only that pressure imparted to it by the blades in that section of the wheel. This produces leakage back into the wheel as soon as the pressure at the discharge builds up due to restriction. Experiment shows that cutoff points with very small clearances have a merit which exists only in fancy. The effect upon efficiency of a reasonably large clearance at this point is negligible; the amount of clearance may be 5 per cent of the wheel diameter in medium-sized fans. Most of the noise in a fan originates at the cutoff, the function and form of which make it like a reed that will give a musical note unless this is guarded against. A blunt, rigid construction as in Fig. 5, will keep the noise down, whereas small clearance and a sharp, knife-like edge will give a loud, shrill noise. In very-high-speed fans the cutoff is sometimes made of a material less subject to vibrations than metal, such as soft wood.

A common form of inlet still used on many casings has a diameter equal to (and in some cases less than) the inlet of the wheel, a casting or rolled angle being used to facilitate connection to a pipe, as in Figs. 7 and 8. With these forms the contraction of the entering stream of air constitutes a restricted area and is a source of loss, as it requires a higher velocity for a given volume of air. The shape of the inlet connection should be conformed to the natural path of the outer particles of air by means of a conical or bell-shaped inlet connection such as shown in Fig. 9. A connection in the form of a frustum of a cone is preferable from a manufacturing standpoint.

The angle of convergence or taper of the inlet cone is best taken at about 15 deg. to the axis (under which condition the coefficient of influx is nearly unity), and should not exceed 30 deg. Its larger diameter should be at least 25 per cent greater than the small diameter (see also Fig. 12, full lines).

Where a fan draws its air direct from the atmosphere or from the room in which it is placed and does not have a pipe attached to its inlet, it is of material advantage to use a fan with two inlets. For a given volume delivered the inlet velocity where two inlets are used will be one-half (and therefore the inlet velocity pressure one-quarter) that with one inlet. The effect of this reduction in inlet losses is illustrated in the table below, where the two fans are identical except as to inlets.

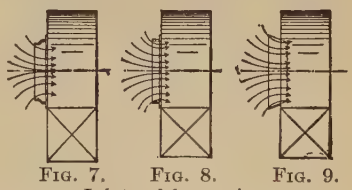


FIG. 7. FIG. 8. FIG. 9.
Inlets of fan casings.

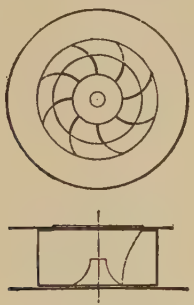


FIG. 10.—Diffuser casing.

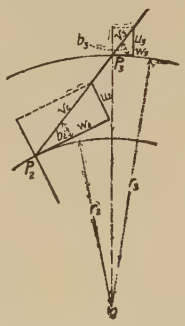


FIG. 11.

Inlet	Revolutions per minute	Static pressure, inches of water	Cubic feet per minute	Air horsepower	Brake horsepower	Mechanical efficiency, per cent
Single.....	265	1.5	39,200	12.50	25.0	50.0
Double.....	265	1.5	44,100	14.95	27.7	54.0

Another test, in which the volume and static pressure were held constant instead of the revolutions per minute and static pressure, shows the decrease in revolutions per minute and increase in mechanical efficiency of the double-inlet fan, as follows [see also Figs. 13 (full lines) and 12]:

Inlet	Revolutions per minute	Static pressure, inches of water	Cubic feet per minute	Air horsepower	Brake horsepower	Mechanical efficiency, per cent
Single.....	265	1.5	39,200	12.50	25.0	50.0
Double.....	253	1.5	39,200	12.50	22.6	55.2

This gain in volumetric and mechanical efficiency holds only with a single-width fan. A double-width, double-inlet fan is in effect simply two single-width, single-inlet fans placed back to back. Obstructions to the flow of air into the inlets, such as bearings and bearing supports, are often important offenders. The inlet area is

reduced and the inlet losses for a given volume increase as the square of the decrease in inlet area due to this cause, while the interference with stream lines adds further to these inlet losses. The table below gives the results of tests on a multiblade type of fan (*a*) with a free and unobstructed inlet and (*b*) with a bearing and its support

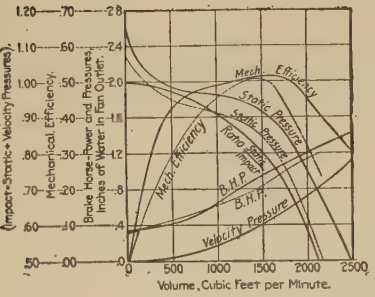


FIG. 12.—Characteristics of a multiblade fan with single-inlet spiral casing.

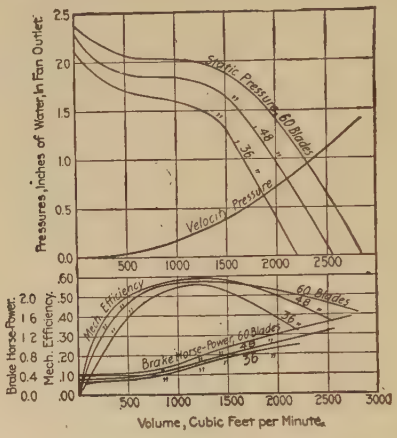


FIG. 13.—Characteristics of multiblade fans with double-inlet spiral casings.

mounted directly in the inlet. The effect on the volumetric and mechanical efficiencies with a given number of revolutions per minute and static pressure is very noticeable.

Test	Revolutions per minute	Pressure, static	Inches of water		Cubic feet per minute	Horsepower		Mechanical efficiency, per cent
			Impact	Velocity		Air	Brake	
(*a*)	900	3.0	4.45	1.45	26,500	18.6	37.0	50.3
(*b*)	900	3.0	3.91	0.91	21,000	12.9	27.0	48.0

Outlet or Discharge of Casing.—The area of the outlet of a spiral casing is seldom made as small as the area between the wheel and spiral at the point of maximum area or where the air passes the cutoff in leaving the casing. A relatively large velocity pressure exists at this point and it is general practice in fans of small and medium size to make no attempt whatever to conserve the kinetic energy in this blast. In most installations this velocity pressure will be from 25 to 50 per cent of the total head developed. Fans in general have outlets varying from 25 per cent to 75 per cent larger than the area at the cutoff point. The expansion is abrupt, so that although some of the velocity pressure is transformed into static pressure, there is nevertheless considerable loss. If the fan discharges directly into the atmosphere the entire velocity pressure in the outlet is lost. Most of this can be saved, however, if the outlet is fitted with a proper evasè discharge piece (see Figs. 13 and 14). In larger fans such as are used for mine ventilation the evasé chimney is started from the cutoff point and thereby conserves the larger portion of the velocity head at that point. If the fan is connected to a duct larger than the outlet it should have a long, tapered connection.

It is not practical to carry the expansion piece on a fan outlet to a point where the velocity is below 1,000 to 500 ft. per minute. The velocity head due to a velocity of 500 ft. per minute, is but $\frac{1}{64}$ in., and is negligible. The angle between the sides and the axis of an efficient chimney or diffuser should be from 7 deg. to 10 deg. The use of a diffuser produces a lower static pressure at the cutoff point and consequently the fan may run at a lower speed and at higher volumetric efficiency. Actual tests on a multiblade fan for mine ventilation showed that, when fitted with a chimney whose larger end was twice the cutoff area and whose sides had a taper of 8 deg. from the axis, the volumetric capacity was increased 12 per cent when operating against a given static pressure and at a given number of revolutions per minute. The horsepower, however, increased only 5 per cent.

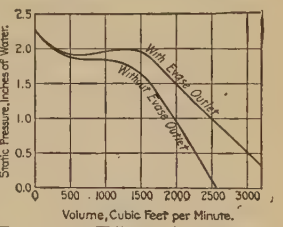


FIG. 14.—Effect of using an evasé discharge on fan.

Diffuser Casing Open Around Circumference.—Similar results may be obtained with wheels discharging into circular diffusers open all around their periphery. Figure 10 shows this form of diffuser, which incidentally has still another

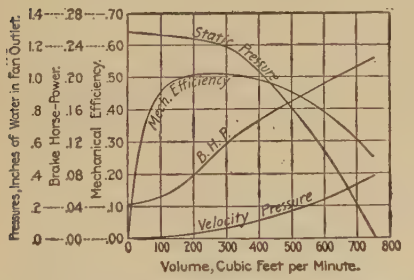


FIG. 15.—Characteristics of a single-inlet steel-plate paddle-wheel fan with radial blades.

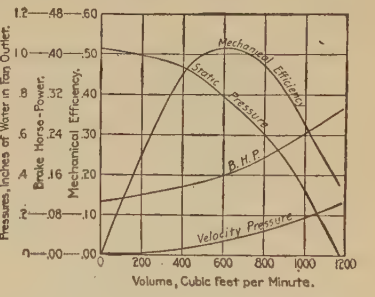


FIG. 16.—Characteristics of a single-inlet fan for high speeds and low pressures.

effect on the outflow of air. This type of fan must deliver its air against the pressure in the chamber into which it discharges, and this pressure acts in a radial direction at all points. It follows, therefore, that the resultant direction of flow should be as near radial as possible. This necessitates a blade curved backward relative to rotation, but it can be considerably helped by a radial diffuser. Figure 11 shows that the velocity of the air decreases in this diffuser and at the same time the direction of the flow becomes more nearly radial; also, that the radial component of the air velocity becomes greater relative to the resultant as the distance from the center increases, thus utilizing a greater part of the resultant velocity. In Fig. 11, r_2 is the radius of the wheel or entrance to diffuser, r_3 the outer radius of diffuser, and A_2 is the circumferential area at entrance to the diffuser. The volume delivered is $u_2 A_2 = v_2 \sin b_2 A_2$. No force acts on the air as it passes through the diffuser and its direction is unchanged, but its radial component varies inversely as the radius, or $u_3/u_2 = r_2/r_3$. The direction of the air leaving the diffuser is known, and its velocity is determinable since u_3 is known. Both v_3 and w_3 decrease much more rapidly than inversely as the radius, and the ratio of

u_3 to v_3 is greater than that of u_2 to v_2. This type of diffuser is generally made stationary and its sides are made radial, converging or diverging. With a slight convergence there can still be an increase in circumferential area with an increase in radius. When the sides diverge the angle of divergence should not exceed from 7 deg. to 10 deg., in order to conserve a full section of flow (see Fig. 18). A diffuser of this sort is also at times applied to a fan wheel running in a spiral casing, the air passing from the wheel to the diffuser and thence to the spiral, which must have small radial depth.

Characteristics of Various Designs and Effect of Change in Elements of Design and Proportions.—The effect of a change in a single element of design on the principal factors of fan performance is seen by comparison of the characteristic curves in Figs. 12 to 18. The curves are drawn for fans handling standard air at a density of 0.075 lb. per cubic foot, having wheels 1 ft. in diameter and operating at a peripheral speed of 4,000 ft. per minute or at 1,274 r.p.m. This facilitates comparison of one with another. The speed of 4,000 ft. per minute is not only a good average speed, but for standard air represents the velocity producing a pressure of 1 in. of water gage. The diameter of 1 ft. was chosen in order to establish as a basis unit diameter and thereby simplify calculations. From these curves the performance at any other peripheral speed can be computed by considering the volume to vary as the first power, the pressures as the second power and the horsepower as the third power of the speed, while the efficiency remains constant for any given load point on the curves. The performance of a symmetrical fan of different size or the computation of size, horsepower, revolutions per minute, etc., necessary to do a given work, is readily accomplished by use of the above rules and the further rule that the capacity of a given design of fan operating against a given static pressure at a given peripheral speed varies as the square of the diameter. All effective fan dimensions will vary as the first power and areas as the second power of the diameter. Where the fan handles air or gas at a density different from standard, the pressure and horsepower will vary directly with the density, and the mechanical efficiency will not be affected. For factors to apply with variations to the air (or gas) temperature, see Table 1, p. 143.

Figure 12 deals with a multiblade type of wheel fitted with a single-inlet spiral casing. The blades are of relatively small radial depth. The curves with solid lines are for blades having radially disposed corrugations equally spaced along the axial length of the blade; the broken-line curves are for similar blades without corrugations. They are curved so that the concave surface moves forward in rotation and are so inclined forward that a chord of the arc of the blade is about 18 deg. ahead of a radius through the tip of the blade, thereby making the outlet ports smaller than the inlet ports of the blades. The principal proportions, given as functions of the wheel diameter, are as follows: Number of blades, 60; radial depth of blades, $0.066D$; axial length of blades, $0.52D$; diameter at inlet of wheel, $0.868D$; equation of spiral, $R = r(1 + 0.198a)$; width of spiral, $0.69D$; number of inlets, 1; diameter of inlet cone, $1.05D$; area of cutoff point, $0.434D^2$; area of outlet, $0.59D^2$.

Method of Using Characteristic Curves.—To illustrate the use of these characteristic curves a problem may be taken, as follows: What will be the size of a fan with corrugated blades of the type covered by Fig. 12 which will be required to deliver 50,000 cu. ft. per minute against 1.5 in. static pressure, the fan to operate at approximately maximum mechanical efficiency? The diameter of the fan found by calculation will vary according to the point on the curves of Fig. 12 taken as a basis, and will

become less as the volume represented by that point increases. Assume as a basis the delivery of a fan 1 ft. in diameter at 51.5 per cent mechanical efficiency where it delivers 1,500 cu. ft. per minute against 1.67 in. static pressure at 1,274 r.p.m., requiring 0.95 b.hp. Operating at the same point on the mechanical efficiency curve but against 1.5 in. pressure would require the speed to be lowered in proportion to the square root of the decrease in pressure, or be multiplied by the factor 0.947. The volume would decrease in the same proportion and the brake horsepower as the third power of 0.947, giving a delivery of 1,412 cu. ft. per minute against 1.5 in. static pressure at 1,200 r.p.m. and requiring 0.81 b.hp. Since a delivery of 50,000 cu. ft. per minute is desired at this static pressure of 1.5 in., the peripheral speed will remain the same but the diameter will vary as the square root of the increase in volume. The speed in revolutions per minute will vary inversely as the diameter and the brake horsepower directly as the volume, giving a fan 5.95 ft. in diameter to deliver 50,000 cu. ft per minute against 1.5 in. at 202 r.p.m. and requiring 28.7 b.hp. Since the diameter obtained is an odd figure, it might be decided to use a fan of even diameter, say 6 ft., in which case its performance would be calculated by considering that a similar fan 1 ft. in diameter at the same mechanical efficiency would deliver a volume in proportion to the square of the diameters, or $\frac{1}{36}$ of 50,000, which is 1,390 cu. ft. per minute, against 1.5 in. pressure. Since the area of outlet is 0.59 sq. ft. the velocity in the outlet is 2,350 ft. per minute and the velocity pressure 0.347 in., hence the ratio of static to impact pressure is then 0.813. Figure 12 contains a curve of this ratio, which facilitates such computations and does away with cut-and-try methods. From Fig. 12 it is seen that when the ratio of static to impact pressure equals 0.813, the 1-ft.-diameter fan delivers 1,465 cu. ft. per minute against 1.69 in. static pressure for 0.93 b.hp. at 1,274 r.p.m. Now to deliver 1,390 cu. ft. per minute against 1.5 in., the fan speed would drop in direct proportion to the volume, or to 1,209 r.p.m., and the brake horsepower as the cube, or to 0.79 b.hp. From this the speed and horsepower of the 6-ft. fan when delivering 50,000 cu. ft. per minute against 1.5 in. are found to be 201 r.p.m. and 28.5 b.hp.

Figure 13 covers a double-inlet fan with wheels and casing of the same proportions as the fan of Fig. 12 the three wheels tested in this casing had respectively 60, 48 and 36 blades. The one with 60 blades was the same wheel as used in Fig. 12. The effect of making the casing with a double inlet was to increase the volumetric capacity and mechanical efficiency by an appreciable amount. The effect of a reduction in the number of blades was a decrease in volumetric capacity and pressure developed, yet the mechanical efficiency increased somewhat with 48 blades. Figure 14 is for the same fan as that having 48 blades in Fig. 13, and shows what an evasé (expanding) discharge on the fan will do in raising the resistance against which a given volume can be delivered or raising the volume delivered against a given resistance. The evasé discharge connection increased in area with a taper to its sides of about 7 deg. until it was twice that of the fan outlet. For any given volume the brake horsepower is the same as for the fan of Fig. 13, the gain being in static pressure produced.

Figure 15 covers a steel-plate paddle-wheel fan with eight flat radial blades of relatively large radial depth and a single-inlet spiral casing. The side plates are inclined inwardly as in Fig. 20. The principal proportions are as follows: Number of blades, 8; radial depth of blades at intake, $0.145D$; maximum radial depth of blades, $0.29D$; axial length of blades (outer edges), $0.39D$; axial length of blades (at intake), $0.46D$; diameter at inlet of wheel, $0.71D$; equation of spiral, $R = r(1 + 0.106a + 0.125)$; width of spiral, $0.5D$; number of inlets, 1; diameter of inlet, $0.71D$; area at cutoff point, $0.306D^2$; area at outlet, $0.306D^2$. While there is not much difference in mechanical efficiency between this fan and the multiblade fan of Fig. 12 there is a great difference in the pressures and volumetric capacities; much less space is required for the multiblade type when performing a given duty.

Figure 16 covers a fan designed for high rotative speeds at relatively low pressures. The wheel is fitted with a single-inlet spiral casing and has 16 deep blades with radially disposed corrugations equally spaced along their axial length. They are so curved that the concave surface moves forward in rotation and are so inclined rearwardly that a chord of the arc of the blade is about 35 deg. back of a radius through the tip of the blade, thereby making the outlet ports larger than the inlet ports of the blades. Reference to Fig. 12 will show the greater peripheral speed required of the fan of Fig. 16 for a given pressure. The principal proportions are as follows: Number of blades, 16; radial depth of blades, $0.135D$; axial length of blades, $0.52D$; diameter inlet of wheel, $0.866D$; equation of spiral, $R = r(1 + 0.198a)$; width of spiral, $0.69D$; number of inlets, 1; diameter of inlet cone, $1.05D$; area at cutoff point, $0.344D^2$; area of outlet, $0.59D^2$.

The full-line curves of Fig. 17 cover a fan designed for high rotative speeds at relatively high pressures such as are found in forced-draft applications. The wheel has no casing and discharges directly into the compartment in which the fan maintains a pressure. The wheel has 16 blades so curved that the convex surface moves forward in rotation. The blade is of variable depth, being most shallow at the intake side of the wheel and deepest at the part farthest from the intake side, forming a type known as a cone fan. The blades have a very decided back slope, which, as previously shown,

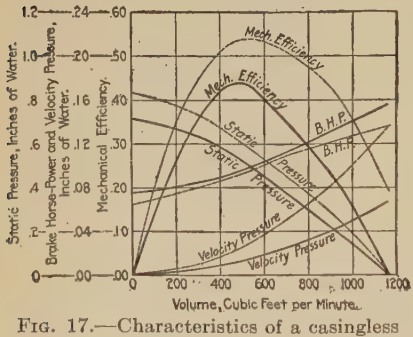


FIG. 17.—Characteristics of a casingless fan for forced-draft applications.

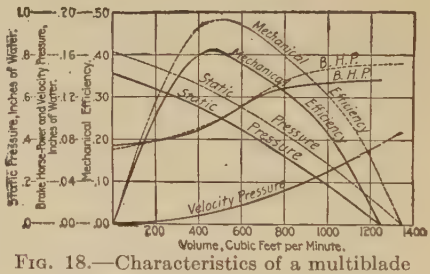


FIG. 18.—Characteristics of a multiblade casingless fan for forced-draft work.

is essential in an open-running wheel. The efficiency curves of Figs 17 and 18 are those obtained by the use of static pressure in computing the air horsepower, since obviously the kinetic energy at discharge from the fan is not useful as in a cased fan where the air generally travels along ducts. The principal proportions are as follows: Number of blades, 16; radial depth of blades, intake side, $0.052D$; radial depth of blades, rear of wheel, $0.28D$; axial length of blades, $0.38D$; diameter of inlet of wheel $0.896D$; outlet port area of wheel, $0.56D^2$. The broken curves of Fig. 17 cover the same fan, except that the wheel is fitted with a single-inlet spiral casing. The volumetric capacity and pressure are increased slightly and the mechanical efficiency considerably; the latter increase being due to the gain in pressure head incident with the expansion from the outlet ports of the wheel into the casing and from the cutoff of the casing to its outlet. This fan is particularly adapted to high-speed drives such as steam turbines. The principal proportions of the casing are as follows: Equation of spiral, $R = r(1 + 0.318)a$; width of spiral, $0.64D$; number of inlets, 1; diameter of inlet cone, $1.2D$; area of cutoff, $0.64D^2$; area of outlet, $0.785D^2$.

Capacity Tables.—The accompanying tables give average performances of fans of the stated types, for a limited number of sizes and for given speeds. The performance at other pressures, speeds and diameters can be computed in the

manner shown on page 154. The approximate height in column 2 is the overall dimension for horizontal discharge.

Multiblade Fans are made by American Blower Co., Sirocco; Buffalo Forge Co., Conoidal; Keith & Blackman, Keith; N. Y. Blower Co., Seri-Vane; B. F. Sturtevant Co., Multivane. The wheels contain from 30 to 60 blades, generally

FIG. 19.—Multiblade fan.

FIG. 20.—Paddle-wheel fan with cast casing.

of shallow radial depth and curved with concave surface forward in rotation. The inlet is exceptionally large compared to the wheel diameter. A spiral casing encloses the wheel (see Fig. 19). The range in pressure is ordinarily 0 to 5 in. of water.

Wheel diameter, feet	Approximate height, feet	Static pressure, inches											
		0.5			1			2			4		
		Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower
1	2	1,170	780	0.30	1,650	1,100	0.85	2,340	1,560	2.4	3,300	2,200	6.8
2	4	4,650	390	1.2	6,600	550	3.4	9,300	780	9.6	13,200	1,100	27.0
4	8	18,700	195	4.8	26,400	275	13.6	37,400	390	38.5	52,800	550	109.0
6	12	42,000	130	10.8	59,400	184	30.5	84,000	260	86.0	118,800	368	244.0
8	16	75,000	98	19.1	105,600	138	54.0	150,000	196	153.0	211,200	276	432.0
10	20	117,000	78	30.0	165,000	110	85.0	234,000	156	240.0	330,000	220	680.0

Cast casing paddle-wheel fans are made by the American Blower Co., Type V; Buffalo Forge Co., Type B; Green Fuel Econ. Co., Cast-iron Volume Fan; N. Y. Blower Co., Peerless; B. F. Sturtevant Co., Monogram.

Wheel diameter, inches	Approximate height, inches	Static pressure, ounces											
		1			2			4			6		
		Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolution per minute	Brake horsepower
10	19	360	2,050	0.18	510	2,900	0.50	720	4,100	1.45	880	5,000	2.65
15	27	810	1,390	0.40	1,150	1,970	1.15	1,620	2,780	3.2	1,980	3,400	5.9
23	42	2,150	885	1.05	3,050	1,250	2.9	4,300	1,770	8.4	5,250	2,170	15.5
33	58	4,650	630	2.3	6,450	890	6.5	9,300	1,260	18.5	11,400	1,540	34.0

The wheels are of paddle-wheel type—6 to 8 blades, and not quite so wide axially as the paddle wheels in "steel-plate" fans. Casings are spiral in form and of cast iron (see Fig. 20.) The range of pressures is from 0 to 6 oz., and they are used for forges, oil furnaces, light dust collecting, etc.

Cast-iron and steel-plate pressure fans are made by the American Blower Co., Type P; Buffalo Forge Co., Type P; B. F. Sturtevant Co., "Steel Pressure Blower." The wheels are of paddle type; number of blades from 6 to 24, width narrower than other types. Casing of spiral form (see Fig. 21). The range of pressures is from 4 to 16 oz. and the application is for forges, oil furnaces, cupolas, gas producers, boosters, etc.

Wheel diameter, feet.	Approximate height inches	Static pressure, ounces											
		4			8			12			16		
		Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower
14	25	600	2,800	1.3	850	3,960	3.7	1,040	4,850	6.7	1,200	5,600	10.4
21	39	1,300	1,900	2.8	1,850	2,680	7.9	2,250	3,300	14.5	2,600	3,800	22.5
34	59	3,400	1,150	7.1	4,800	1,625	20.0	5,900	2,000	37.0	6,800	2,300	57.0
45	77	6,300	875	13.0	8,900	1,240	37.0	10,900	1,500	67.0	12,600	1,750	104.0

Steel-plate planing-mill fans (slow-speed type) are made by the American Blower Co., Buffalo Forge Co., Green Fuel Econ. Co., Sterling Blower Co., B. F. Sturtevant Co., etc. The wheels have 12 to 18 paddle-type blades, curved forward. Casing is of spiral form, about the same as in common steel-plate fans (see Fig. 22). The range of pressures from 0 to 6 oz. and they are used for conveying materials such as refuse from wood-working machinery, shoe machinery, buffing and emery wheels, etc.

Wheel diameter, inches	Approximate height, inches	Static pressure, ounces											
		1			2			4			6		
		Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower
24	35	1,770	650	0.7	2,500	920	2.0	3,540	1,300	5.6	4,350	1,600	10.2
36	50	4,000	435	1.6	5,600	615	4.5	8,000	870	12.8	9,800	1,060	23.5
48	70	7,050	325	2.8	10,000	460	8.0	14,100	650	22.4	17,300	800	41.0
64	90	12,600	245	5.0	17,800	345	14.2	25,200	490	40.0	31,000	600	73.0

Steel-plate paddle-wheel fans are made by the American Blower Co., Buffalo Forge Co., Garden City Fan Co., Green Fuel Econ. Co., N. Y. Blower Co., B. F. Sturtevant Co., etc. The wheels have 8 to 12 blades which are deep radially and

connected to spider arms emanating from a central hub. Casings are of spiral form (see Fig. 20). The range of pressures is from 0 to 5 in., and they are used for forced draft, induced draft, heating, ventilating, cooling, drying, etc.

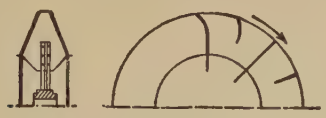


FIG. 21.—Cast-iron or steel-plate pressure fan.

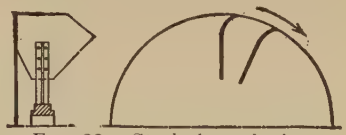


FIG. 22.—Steel-plate planing-mill fan.

Wheel diameter, feet	Approximate height, inches	Static pressure, inches											
		0.5			1			2			4		
		Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower
2	40	1,800	510	0.37	2,500	720	1.1	3,600	1,020	3.0	5,000	1,440	8.6
3	60	4,000	340	0.85	5,700	480	2.4	8,000	680	6.8	11,400	960	19.0
4	80	7,200	255	1.5	10,200	360	4.3	14,500	510	12.0	20,400	720	34.0
6	120	16,000	170	3.4	22,800	240	9.6	32,000	340	27.0	45,600	480	77.0
8	160	28,700	128	6.0	40,700	180	17.1	57,500	255	48.0	81,400	360	137.0
12	240	65,000	85	13.5	91,200	120	38.4	130,000	170	108.0	182,000	240	307.0

Steel-plate cone fans are made by the American Blower Co., Buffalo Forge Co., Green Fuel Econ. Co., N. Y. Blower Co., B. F. Sturtevant Co. These fans are of the paddle-wheel type, but built on a back or supporting cone instead of a spider having arms emanating from a central hub; 8 to 12 blades are employed and no casing is used. The fan discharges directly into the atmosphere or a large room, as the case may be (see Fig. 23). The pressures run from 0 to 1½ in. and the fans are used for heating, ventilating, cooling, drying, etc.

FIG. 23.—Steel-plate cone fan.

Wheel diameter, feet	Static pressure, inches											
	0.25			0.5			1.0			1.5		
	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolution per minute	Brake horsepower
3	4,300	320	0.63	6,100	450	1.8	8,600	640	5.0	10,400	785	9.2
4	7,650	240	1.1	10,800	340	3.2	15,300	480	8.9	18,700	590	16.3
6	17,200	160	2.5	24,300	225	7.1	34,400	320	20.0	42,000	390	37.0
8	30,500	120	4.5	43,000	170	12.7	61,000	240	36.0	75,000	295	66.0
12	69,000	80	10.0	98,000	113	28.0	138,000	160	80.0	169,000	195	147.0

In general I have found that after a few months service most fans give materially less than the maker's rating and allowance should be made for this in buying a fan.

Steam-jet Blowers.—Steam jets have long been used for "blowing" or exhausting in order to maintain combustion in locomotive boilers, usually employing the exhaust from the engines through properly shaped "nozzles." This type of air compressor or exhauster also finds extended application for emergency use and at times in permanent installations for removing foul air from mines, factories, ship holds, and for gas exhausters, for securing forced draft, and for handling gases under low pressures in certain chemical industries. Among its advantages are simplicity, ease of operation, small space, minimum of repairs and ease of regulation.

A. von Ihering ("Die Gebläse") reports some tests as to steam consumption which indicate from 1.3 to 3.4 lb. of steam required per 1,000 cu. ft. of air handled, the larger sizes being the more economical.

Hydraulic Compressors.—Several devices have been made for utilizing falling water for the purpose of compressing air without the use of any mechanical moving parts. The most successful of these is the Taylor compressor, shown diagrammatically in Fig. 24. In the figure, air tubes are represented at *C*, all terminating at the conical entrance *B* to the down-flow pipe *E*. The water supply is furnished through the flume *D*. As the water falls it draws air through the small tubes, carrying it down to the separating tank *G*, where it is liberated at a pressure depending on the weight of water in the vertical pipe *H*. The compressed air is then conducted through the pipe *K* to the place where it is to be used. The distance from *M* to the tail race *L* represents the height or fall of water that is available.

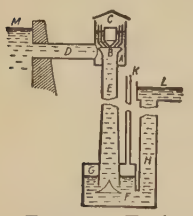


FIG. 24.—Taylor hydraulic air compressor.

In this system the compression is isothermal and the compressed air is saturated with moisture. The oxygen content of the air is reduced, which renders the air less beneficial for purposes of mine ventilation if the exhaust from the air tools is planned to assist ventilation. The system offers a very simple solution for utilizing water powers when the market for compressed air justified its installation. It has the advantage of simplicity with a minimum of operating expense, and very high efficiencies are secured. The first cost of the installation is likely to be high.

Air measurements, absolute pressures				Water measurements			
Free air, cubic feet per minute	Free air, pounds per square inch	Compressed air, pounds per square inch	Horsepower	Cubic feet per minute	Head, feet	Horsepower	Efficiency, per cent
10,580	14	128	1,430	13,057	70.5	1,741	82.17
11,930	14	128	1,623	14,820	70.0	1,961	82,27
9,238	14	128	1,248	12,710	70.6	1,700	73.50

Figure 25 illustrates some of the dimensions of a Taylor hydraulic compressor installed near Cobalt, Ontario, Canada. This was designed for a capacity of 40,000 cu. ft. of free air per minute to be compressed to a gage pressuae of 120 lb. The compressed air is conducted to mines through 9 miles of 20-in. pipe leading to two 12-in. lines with a total distributing line of 21 miles in length. The water is admitted

through suitable gates to two "heads" each 16 ft. in diameter and containing 66 pipes 14 in. in diameter. The size of the heads is reduced in diameter to about 8 ft. and the whole apparatus can be raised or lowered as required by operating conditions. A cone *a* assists in separating the air and water, and the long horizontal tunnel permits quite complete separation. The compressed air is removed through the pipe *c* and the water freed from the entrained air escapes through the vertical shaft *b*. Pipe *e* acts as a relief for a surplus of compressed air. Its end is normally below the surface of the water in the tunnel, but if too much air should accumulate the end would be exposed and permit the escape of the surplus without seriously affecting the normal air pressure of the distribution system.

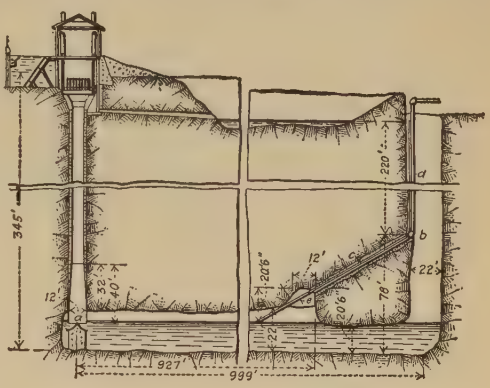


FIG. 25.—Taylor hydraulic compressor installation near Cobalt, Ont.

Piston Compressors and Blowers.—The large quantities of air required for blast-furnaces and Bessemer converters are usually supplied by piston compressors of large capacity, driven either by steam or gas engines. Turbo-blowers directly driven by steam turbines, however, have been recently developed for this work.

Piston compressors used as blowers for blast furnaces and Bessemer converters usually operate at discharge pressures of 15 or 20 lb. gage. But small compressors of this type are built for pressures as low as 1 lb. above atmosphere, single-stage up to 80 to 100 lb., two-stage from 80 to 500 lb., three-stage to about 1,200 lb. (small units even to 2,500 lb.) and four-stage to 3,000 lb. or even higher. The tendency in blower design is to secure increased capacity by higher speeds than formerly. The Allis-Chalmers Company now uses a maximum speed of 90 r.p.m. for 48-in. stroke and 85 r.p.m. for 60-in. stroke, giving piston speeds of from 720 to 850 ft. per minute. With piston speeds approximating 750 ft. per minute the inlet area is approximately 13 per cent and the outlet area 11 per cent of the piston area. With increased speeds the tendency is to increase clearance, which will usually vary 9½ to 11½ per cent.

The Slick blowing tub consists of a reciprocating cylinder on the outside of the compressing cylinder, arranged so as to open ports at the ends of the cylinders for inlet. This gives unobstructed inlet areas of from 18 to 20 per cent with very small clearance.

The ratio of inlet valve area to piston area varies from 0.05 to 0.14. For ordinary types of valves, the inlet area should, as a rule, be not less than 8 or 10 per cent. of the piston area. Automatically operated inlet valves are apt to be irregular in their action and reduce the volumetric efficiency of the compressor. This disadvantage is overcome by using mechanically operated inlet valves, usually of a Corliss type, but on account of the variation of discharge pressure they are not used so often for discharge or for inlet on the high-pressure stages of a multi-stage compressor. The area of the discharge valves will usually vary from 10 to 15 per cent of the piston area, the larger percentage being required for the higher piston speeds.

In the Laidlaw-Dunn-Gordon compressors a valve gear is used that mechanically controls the openings and closing of the suction and the closing of the discharge.

The opening of the discharge is effected by means of vertically placed poppet valves. At the end of each stroke, however, the discharge is mechanically closed by means of a Corliss semi-rotary valve directly under the poppet valves, thus retaining under the latter a cushion of air under discharge pressure, enabling them to seat without violent and noisy impact. The construction is such that very small clearances are possible. In a 28 × 30-in. cylinder, for example, it is but 0.7 per cent. This type of gear has proved very successful on dry air pumps when very high vacuum is required.

AIR COMPRESSION

Theory of Air Compression.—If it be assumed that a compressor works under ideal conditions without clearance and without friction losses and the compression from p_1 to p_2 (Fig. 26) follows the law $pv^n = K$, the work done is $W = n(p_2v_2 - p_1v_1)/(n-1) = np_1v_1[(p_2/p_1)^{(n-1)/n} - 1]/n - 1)$. The temperature at the end of the compression is given by $T_2/T_1 = p_2/p_1{}^{(n-1)/n}$. The work W

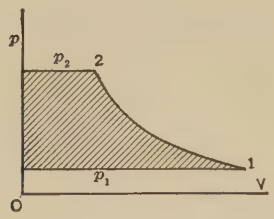


Fig. 26.

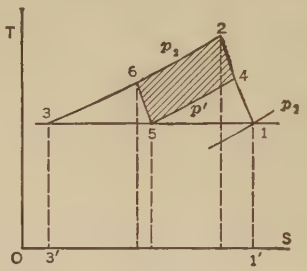


Fig. 26a.

Air-compressor cycle.

decreases as n decreases and to decrease n from its adiabatic maximum ($n = \gamma = 1.406$ approximately) is the function of the water jacket (or the petticoats on the cylinder, if air-cooled).

When the pressure p_2 is high it is advantageous to divide the process into two or more stages and cool between the cylinders. The saving is best shown on the temperature-entropy diagram (Fig. 26*a*). With single-stage compression, 12 represents the compression from p_1 to p_2, and if the constant-pressure line 23 is drawn cutting the isothermal through point 1 in point 3, the area 1′1233′ represents the work W. When two stages are used, 14 represents the compression from p_1 to an intermediate pressure p', 45 cooling at constant pressure in the intercooler between the cylinders, and 56 the compression in the second stage. The area under 14563 represents the total work in the two stages and 2456 the saving due to compounding. The saving is at a maximum when $T_4 = T_6$, which is the case when the intermediate pressure $p' = \sqrt{p_1p_2}$.

The total work in two-stage compression is $np_1v_1[(p^1/p_1)^{(n-1)/n} + (p_2/p')^{(n-1)/n} - 2]/n-1$.

Mean Effective Pressure in Multi-stage Compression.—The mean effective pressure, pounds per square inch, with complete intercooling, referred to the l.-p. piston, is expressed by the formula

$$p_m = p_1\left\{\frac{Sn}{n-1}\left(\left[\frac{p_2}{p_1}\right]^{\frac{n-1}{Sn}} - 1\right]\right\}$$

where S represents the number of stages and n is the exponent of the compression curve. Figure 27, from a chart plotted by F. W. O'Neill, shows the relation between the mean

effective pressure and the initial pressure for various pressure ratios, p_2/p_1, with adiabatic compression and complete intercooling.

The ideal method of compressing air when it is to be stored or allowed to cool before being used is the isothermal, and in the earlier types of compressors this was attempted by so-called wet compressors, by means of which it was possible to secure compressions approximating $p_1v_1^{1.2} = p_2v_2^{1.2}$. The mechanical difficulties involved and the necessary low speeds with consequent small capacity have led to the use of modern "dry compressors," which in small sizes have cylinders with cast-iron ribs for radiating heat, and in large sizes have water jackets surrounding the cylinder. The cooling thus secured is sufficient to keep temperatures from being excessive, but as a rule the compression curves are above $pv^{1.3} =$ constant. Dry compression has the advantage of higher speeds and larger capacities.

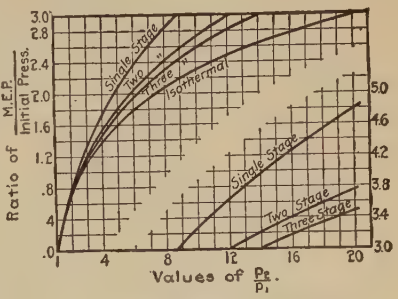


FIG. 27.—Mean effective pressures in air compression.

Efficiencies.—The effect of clearance upon capacity is usually expressed in terms of *volumetric efficiency.*

The *apparent volumetric efficiency* is the apparent volume of free air drawn in (as shown by the indicator card) divided by the volume of the piston displacement. This is the term that is commonly used in speaking of volumetric efficiency, and in Fig. 29 it is GK/L. If the clearance expansion line follows the equation $p_1v_1^n = p_2v_2^n$, where the clearance $C = v_1/L$, then $v_2/L = (p_1/p_2)^{\frac{1}{n}}C$. The volumetric efficiency may also be written:

$$1 - \frac{v_2 - v_1}{L} = 1 - C\left[\left(\frac{p_1}{p_2}\right)^{\frac{1}{n}} - 1\right].$$

Example.—If p_1 is 94.7 lb. per square inch absolute and C is 2 per cent., the volumetric efficiency will be $1 - 0.02\left[\left(\frac{94.7}{14.7}\right)^{\frac{1}{1.4}} - 1\right] = 0.9444$.

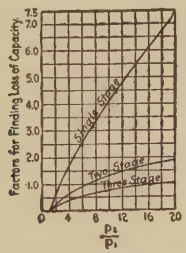


FIG. 28.—Air compressor card.

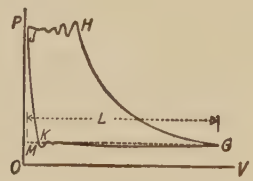


FIG. 29.

The loss in capacity for stage compression will be represented by the following formulas in which p_1 is the initial and p_2 the final pressure:

Two-stage: $1 - C\left[\left(\frac{p_2}{p_1}\right)^{\frac{1}{2\times1.4}} - 1\right]$; Three-stage: $1 - C\left[\left(\frac{p_2}{p_1}\right)^{\frac{1}{3\times1.4}} - 1\right]$

Figure 29 represents graphically the part within the brackets of the above equations. Knowing the actual clearance and the pressure range, the effect

of this clearance on capacity can be found. For example, with a pressure ratio of $p_2/p_1 = 8$, the chart shows this value for single-stage compression to be 3.3. The capacity for these conditions and a 4 per cent clearance will be $1 - (0.04 \times 3.3) = 1 - 0.132$, or 86.8 per cent of the piston displacement. Clearances in the larger sizes of compressors approximate 1 per cent; in smaller machines they are greater, being in some cases as high as 3 per cent.

The indicator card would be a true method of measuring the volumetric efficiency if the temperature of the air after being drawn into the cylinder were the same as that of the atmosphere, and if the pressure at the end of the suction stroke were the same as that of the atmosphere. This is never the case.

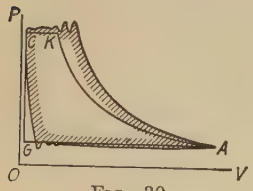


FIG. 30.

The true volumetric efficiency is the ratio of the free air actually drawn in to the piston displacement.

The cylinder efficiency of an air compressor may be defined as the ratio of the work done in a complete cycle to compress isothermally a volume of air at atmospheric pressure equal to the intake piston displacement, divided by the actual work done in the air cylinder. This would be the area *AKCG* (Fig. 30) divided by the shaded area or the actual work done in the air cylinder.

The *efficiency of compression* may be defined as the product of the cylinder efficiency and the true volumetric efficiency, or it is the work done in a complete cycle to compress isothermally (without clearance) a given volume of free air, divided by the work actually expended in compressing the same volume of free air.

The *mechanical efficiency* of an air compressor is the work done in the air cylinders divided by the work done in the engine cylinders if driven direct by steam or gas engine, or by the work delivered at the belt if the compressor is belt-driven.

Actual Values of Efficiencies.—Tests of piston compressors show extreme variations of mechanical efficiency from 76 to 97 per cent, with approximate averages for the more common sizes of 85 per cent. The true volumetric efficiency of piston air compressors will vary from 80 to 97 per cent, and the cylinder efficiency for water-jacketed compressors from 80 to 85 per cent. This will result in efficiencies of compression varying from 64 to 82 per cent.

Multiplying the mechanical efficiency by the efficiency of compression will show variations from 48 to 79 per cent. That is, the energy required to compress a certain amount of air isothermally is only from 48 to 79 per cent of that actually expended in the steam cylinder. Tests show that with the great majority of compressors this product ranges from 50 to 60 per cent, while with some of the best compressors under test conditions it will reach as high as 78 per cent.

Horsepower Required to Compress Air.—Disregarding clearance, the horsepower required to compress air in a single-stage compressor is:

$$\text{horsepower} = \frac{144}{33{,}000} \frac{n}{n-1} p_1 v_1 \left[\left(\frac{p_2}{p_1} \right)^{\frac{n-1}{n}} - 1 \right]$$

where v_1 represents the volume (cu. ft.) of free air compressed per minute and p_1 and p_2 the pressures in pound per square inch absolute.

Representing the intercooler pressure by p_i, the work done in both cylinders of a two-stage compressor will be

$$\text{horsepower} = \frac{144}{33{,}000} \frac{n}{n-1} p_1 v_1 \left[\left(\frac{p_i}{p_1} \right)^{\frac{n-1}{n}} + \left(\frac{p_2}{p_i} \right)^{\frac{n-1}{n}} - 2 \right]$$

With perfect intercooling, $p_1 v_1 = p_i v_i$. The above expression for the total work will be a minimum when $p_i = \sqrt{p_1 p_2}$ as already discussed.

The proper intercooler pressures for three-stage compression are:
First intercooler, $p = \sqrt[3]{p_1^2 p_2}$; second intercooler, $p = \sqrt[3]{p_1 p_2^2}$
The minimum work done in compressing air is given by

$$\text{horsepower} = 2 \times \frac{144}{33{,}000} \frac{n}{n-1} \cdot p_1 v_1 \cdot \left[\left(\frac{p_2}{p_1} \right)^{\frac{(n-1)}{2n}} - 1 \right] \text{ for two-stage compressors,}$$

$$\text{and horsepower} = 3 \times \frac{144}{33{,}000} \frac{n}{n-1} \cdot p_1 v_1 \cdot \left[\left(\frac{p_2}{p_1} \right)^{\frac{(n-1)}{3n}} - 1 \right] \text{ for three-stage compressors.}$$

With perfect intercooling the volumes of the cylinders should be inversely as the pressures of the air admitted to them.

Professor Lucke ("Engineering Thermodynamics") gives a number of charts for solving graphically many of the problems of air compression, of which Fig. 31 is a

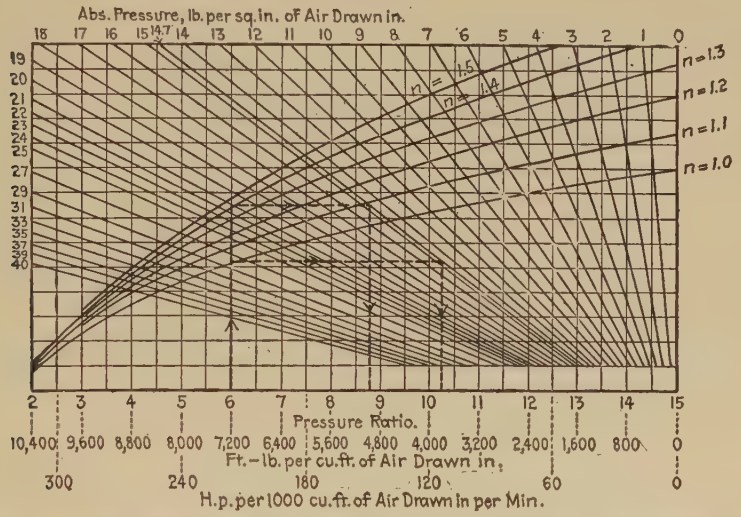


FIG. 31.—Chart for determining the work done in single-stage air compression (Lucke).

very convenient one for determining the work per cubic foot and the horsepower per 1,000 cu. ft. of free air compressed in a single stage to any pressure up to 15 times its initial pressure and for various kinds of compression curves from the isothermal to the adiabatic.

In Fig. 31 the diagonal lines represent various absolute pressures for the free air drawn in and the curved lines apply to various kinds of compression curves. The lower horizontal scale gives pressure ratios, work and horsepower. In using the curves follow vertically from the pressure ratio to the n curve, horizontally to the inlet pressure line and vertically downward to the horizontal axis where the work and horsepower may be read. For example, if the compression ratio is 6, the compression curve follows the equation $p_1 v_1^{1.4} = p_2 v_2^{1.4}$, and the free air is at 14.7 lb. per square inch absolute, there will be required 4,960 ft.-lb. of work per cubic foot of free air compressed, or 152 hp. per 1,000 cu. ft. of free air per minute.

If the compression for the same pressure range (or to 88.2 lb. per square inch absolute) had followed the isothermal compression curve it would have required 3,880 ft.-lb. of work per cubic foot, or 120 hp. per 1,000 cu. ft. per minute.

Figure 31 enables calculations to be made for work and horsepower for two- and three-stage compression, when used in connection with Fig. 31. The dotted lines represent two-stage and the full lines three-stage compression, and are marked according

to the character of the compression curve. The horizontal scale shows pressure ratios and the vertical scale ratio of work or horsepower for two- or three- stage compression. to the work that would be required for single-stage compression as determined from Fig. 31. For example, for a pressure ratio of 8, or a discharge of 117.6 lb. per square inch absolute on a suction pressure of 14.7 lb., the work for a compression following the equation $p_1v_1^{1.4} = p_2v_2^{1.4}$ would be 85.2 per cent of the work for the same conditions single-stage if two-stage were used, and 81 per cent of it if three-stage compression were used.

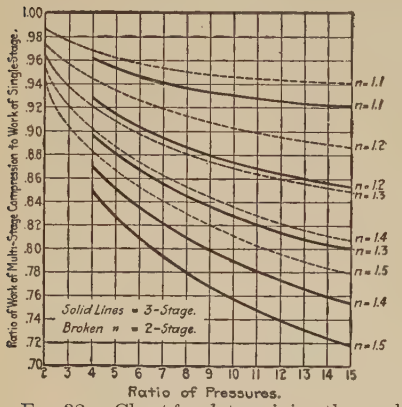


FIG. 32.—Chart for determining the work done in two- and three-stage air compression (Lucke).

Effect of Altitude.—As the density of the atmosphere decreases with the altitude, a compressor located at a high altitude will take in a smaller weight of air at each stroke. The reduction of pressure at the inlet affects the power expended in compressing the air, but the decrease in power required does not vary in the same ratio as the decrease in capacity. For this reason compressors to be used at high altitudes should have the steam and air cylinders properly proportioned to meet the varying conditions at different levels. Table 12, published by the Sullivan Machinery Co., of Chicago,

TABLE 12.—VOLUMETRIC AND HORSEPOWER COEFFICIENTS FOR TWO-STAGE AIR COMPRESSION

Altitude, feet	Barometer pressure, pound per square inch	Terminal gage pressure, pounds per square inch															
		70		80		90		100		120		140		150			
		Horsepower coefficients	Volumetric coefficients	Horsepower coefficients	Volumetric coefficients	Horsepower coefficients	Volumetric coefficients	Horsepower coefficients	Volumetric coefficients	Horsepower coefficients	Volumetric coefficients	Horsepower coefficients	Volumetric coefficients	Horsepower coefficients	Volumetric coefficients		
Sea level	14.72	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00		
1,000	14.17	0.98	0.97	0.98	0.97	0.98	0.97	0.98	0.97	0.98	0.97	0.98	0.97	0.98	0.97		
2,000	13.64	0.97	0.94	0.96	0.94	0.96	0.94	0.96	0.94	0.96	0.93	0.96	0.93	0.96	0.93		
3,000	13.13	0.95	0.91	0.95	0.91	0.94	0.91	0.94	0.91	0.94	0.90	0.94	0.90	0.94	0.90		
4,000	12.64	0.93	0.88	0.93	0.88	0.93	0.88	0.92	0.88	0.92	0.87	0.92	0.87	0.92	0.87		
5,000	12.17	0.91	0.85	0.91	0.85	0.91	0.85	0.91	0.84	0.90	0.84	0.90	0.84	0.90	0.84		
6,000	11.71	0.90	0.82	0.89	0.82	0.89	0.82	0.89	0.82	0.88	0.82	0.88	0.81	0.88	0.81		
7,000	11.27	0.88	0.80	0.88	0.79	0.87	0.79	0.87	0.79	0.86	0.79	0.86	0.78	0.86	0.78		
8,000	10.85	0.86	0.77	0.86	0.77	0.85	0.77	0.85	0.76	0.85	0.76	0.84	0.76	0.84	0.76		
9,000	10.45	0.85	0.75	0.84	0.74	0.84	0.74	0.83	0.74	0.83	0.73	0.82	0.73	0.82	0.73		
10,000	10.06	0.83	0.72	0.83	0.72	0.82	0.72	0.82	0.71	0.81	0.71	0.81	0.71	0.80	0.70		
11,000	9.69	0.82	0.70	0.81	0.70	0.80	0.69	0.80	0.69	0.79	0.68	0.79	0.68	0.79	0.68		
12,000	9.33	0.80	0.68	0.79	0.67	0.79	0.67	0.78	0.67	0.78	0.66	0.77	0.66	0.77	0.66		
13,000	8.98	0.78	0.65	0.78	0.65	0.77	0.65	0.77	0.64	0.76	0.64	0.75	0.63	0.75	0.63		
14,000	8.64	0.77	0.63	0.76	0.63	0.76	0.62	0.75	0.62	0.74	0.62	0.74	0.61	0.74	0.61		
15,000	8.32	0.75	0.61	0.74	0.61	0.74	0.60	0.74	0.60	0.73	0.59	0.72	0.59	0.72	0.59		

Ill., shows the variation in capacity and horsepower for various altitudes. The altitudes given are heights above mean sea level and are subject to correction for temperature and latitude. From the table it can be seen that for a two-stage compressor discharging at 100 lb. pressure when operating at an altitude of 8,000 ft. the volumetric capacity will be only 76 per cent of that at mean sea level, while the horsepower required will be 85 per cent of that at mean sea level.

REGULATION, REHEATING, LUBRICATION

Unloading Devices.—Many compressors operate at constant speed, independent of the demands for compressed air; and, in order to secure economy of operation for this condition, various types of "unloaders" have been designed. For small single-stage compressors the Sullivan Machinery Co. provides an un-

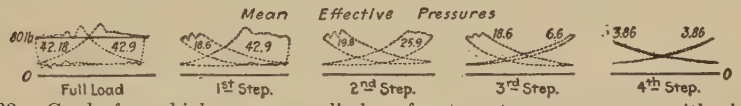


FIG. 33.—Cards from high-pressure cylinder of a two-stage compressor with clearance unloader.

loading valve connected by piping to the inlet valves. When the predetermined pressure is exceeded, the unloader raises the inlet valves from their seats and prevents further compression of air until the pressure falls a few pounds, when the unloader allows the valves to resume their seats and the work of compression is again taken up. Other manufacturers use a similar device to keep the inlet closed when the predetermined pressure is reached. For larger compressors a double-beat valve is used, which is placed on the air inlet duct and controlled by air pressure from the air receiver. This valve is set to shut off from the compressor all or part of the incoming air when the receiver pressure rises above predetermined point.

When the speed is not constant other types of regulating devices are available. For simple belt-driven compressors a belt shifter can be designed to shift the belt from the tight to the loose pulley when the predetermined air pressure is reached, but for steam-driven compressors the device is usually one for governing the speed of the compressor to suit the demands for compressed air.

Water Jackets, Intercoolers, Receivers and Aftercoolers.—The cooling surface in an intercooler is generally designed from the formula $S = Q/0.25(t_a - t_w)$, where S is the cooling surface in square feet, Q is the number of cubic feet of free air per minute, and t_a and t_w are the temperatures, in degrees F. of the air leaving and the water entering the intercooler, respectively.

The amount of cooling water required is given by Longacre as $G = 75 + 2.5t$ when the intercooler and jacket are in series; $G = 20 + 2t$ for a separate intercooler; and $G = 2t$ for low-pressure and high-pressure jackets only. In these formulas G is in gallons of water per hour per 100 cu. ft. of free air per minute and t is the entering temperature, degrees F. of the cooling water.

Receivers are used to supply a reservoir of air; to equalize the pulsations in the air coming from the compressor; to collect the water and grease held in suspension by the compressed air as it leaves the compressor; to reduce the friction of air in the pipe system; and to cool the air as thoroughly as possible before entering the transmission system. To facilitate the removal of water from the

compressed air, a receiver is frequently equipped with a coil of pipes through which cooling water flows; in this way it serves as an aftercooler, and by precipitating the water from the air reduces difficulties in transmission lines and tools. The Ingersoll-Rand Co. states the amount of cooling water required for aftercoolers for air at 80 to 100 lb. pressure to be as follows:

Temperature of cooling water, degree F.	50	60	70	80	90
Gallon per minute per 1,000 cu. ft. of free air per minute	120	140	160	189	200

When the transmission pipe line is long, receivers should be placed at both ends of the pipe.

Reheaters.—Heating the air just before expansion may increase the efficiency of the system, and in addition will increase the temperature at the end of expansion and prevent the freezing up of the motor. In quarry work stoves are sometimes used for preheating the air. In locomotive work for mines and surface use, hot water is frequently employed for this purpose. Reheaters are usually capable of raising the temperature of the air to from 300 to 500°F., although common practice shows temperatures of from 250 to 350°. In figuring on reheaters it is usual to assume that 1 lb. of coal will give from 8,000 to 10,000 B.t.u. to the air. As the specific heat of air is approximately 0.24, 1 lb. of coal will raise the temperature of approximately 100 lb. (or 1,200 cu. ft.) of free air 300°.

The increase in efficiency resulting from reheating is greater with tools that use air expansively than with machines taking in air for full stroke. Sometimes it is not desirable to have the air entering a tool at a temperature above 300°, because of the effect of this high temperature on the lubrication. For these conditions small portable hot-water stove-type reheaters are available in capacities of from 62 to 800 cu. ft. of air per minute.

Lubrication.—If oil is fed too rapidly in the air cylinders there is a gradual accumulation of carbon, which interferes with the free movement of the valves and may actually choke the passages and produce high temperatures sufficient to produce ignition or explosion. Explosions have taken place from the introduction of kerosene or naphtha into the air cylinder for the purpose of cleaning the valves and cutting away the carbon deposits. This is a very effective way of cleaning valves and pipes, but is a source of danger and should be absolutely prohibited. Soft soap and water is the best cleanser for the air cylinder and is recommended even when the best grades of cylinder oil are used, feeding once or twice a week in order to prevent any gumming of the valves.

In order to reduce the danger of excessive temperatures, fusible safety alarm plugs may be inserted in the discharge line. These are usually set for a temperature of 350° for a single-stage compressor working at 40 lb. gage pressure, for a two-stage compressor at 100 lb. gage, and for a three- or four-stage compressor delivering at 1,000 lb. gage. A 500° plug is furnished for use with a single-stage compressor discharging at 100 lb. gage pressure.

Measurement of Discharge.—The relation between the piston displacement and the amount of air actually delivered, has been taken, in the past, from indicator cards, but this method is extremely inaccurate, since slippage due to leakage past the piston rings and leakage back into the cylinder through discharge valves, cannot be detected. As a matter of fact, these leakages tend to make an apparently better card and to increase the apparent volumetric efficiency. Recent

tests have been made by discharging the compressed air through orifices and calculating the quantity of free air delivered per minute by Fliegner's formula.

Fleigner's formula may be stated as follows: $G = (0.53AP)/(\sqrt{T})$, where G = flow in pounds per second, A = area of orifice in square inches, P = absolute pressure (in pound per square inch) of air behind the orifice, and T = absolute temperature (degree Fahrenheit) of the air behind the orifice. The weight of 1 cu. ft. of air is found by the following formula: $W = 1.325B/T$, where W = weight of 1 cu. ft. of air, B = barometer reading in inches of mercury, and T = absolute temperature (degree Fahrenheit) at the compressor intake.

The delivery of cubic feet of free air per minute then equals $G \times (60/W)$.

Efficiency.—In practice it is found that compressors with mechanically operated rotary inlet valves show volumetric efficiency varying from 91 per cent at 100 r.p.m. to 88 per cent at 188 r.p.m. Piston inlet machines at 100 r.p.m. give 88 per cent efficiency, and at 188 r.p.m., 79 per cent.

Centrifugal Compressors.—Centrifugal compressors differ from centrifugal pumps (see p. 114) only in handling gases instead of liquids, and are similarly classified as regards the number of inlets per impeller, the direction of the impeller tips at impeller exit, and the number of stages employed. They are further classified into low-pressure (1 to 5 lb. per square inch) and high-pressure (above 5 lb.) compressors, and also into the radial-inlet type and the axial-inlet type according as the gas enters at right angles to the shaft or in a direction parallel to the shaft.

Centrifugal compressors for pressures below 1 lb. per square inch are generally known as blowers or centrifugal fans (see p. 149); in these the kinetic energy of the gas at the impeller exit is usually allowed to dissipate itself in eddies. For air pressures of 5 lb. per square inch and under, a single impeller is generally sufficient. For comparatively light gases, however, a pressure of 5 lb. may require two or more impellers in series, or a multi-stage compressor. Such a compressor is also frequently spoken of as a high-pressure gas compressor. For quantities of gas of 10,000 cu. ft. per minute and over, compressors of the radial-inlet type require shrouds or reinforcing rings at the inner ends of the impeller blades to prevent the wide blades from crumpling at the inlet under the action of centrifugal stresses. Impellers of the axial-inlet type are not subject to such crumpling, and are therefore generally used when large volumes are handled. As to single-inlet and double-inlet impellers, see remarks on page 161. The radial-discharge impeller is the one best adapted for high peripheral speeds, and is the type most commonly used. Both the backward-discharge and the forward-discharge impellers require shrouds at their outer peripheries. The former are frequently resorted to when large gas quantities (requiring a large impeller inlet) are to be raised to a comparatively low pressure with a direct-connected high-revolution-per-minute driver; the latter are but rarely used. Multi-stage compressors are usually provided with special means for cooling the gas during its passage through each impeller and from stage to stage, and also for preventing leakage from stage to stage and to the atmosphere.

The centrifugal compressor occupies comparatively little room for its output; direct-connected to an electric motor or to a steam turbine, it forms a very compact unit. Besides its bearings it has no rubbing or wearing parts; it contains no moving valves or springs; and it requires a minimum of attendance and oiling. It is also fairly free from vibrations and requires comparatively light foundations. At constant revolutions per minute it will maintain approximately constant pressure for widely varying quantities of gas, which makes it very desirable for general power transmission. If no gas is required temporarily the discharge pipe may be shut off without

stopping the compressor or wasting the gas into the atmosphere. Its steadiness of blast makes it also very valuable for oil burning and for general forge work.

For blast furnaces, where the resistance to the flow of the air is likely to vary from time to time, while a uniform supply of oxygen or air regardless of such variation in resistance is important, the centrifugal compressor may be supplied with a constant-volume governor. This governor, actuated by the variation of air velocity in the compressor inlet, causes the speed of the driver to vary in accordance with the needs of the furnace, and thus maintains a constant volume (referred to atmospheric conditions) of air against widely varying pressures with fairly constant efficiency. The centrifugal compressor also finds wide application as an exhauster in ash conveying, sawdust conveying, and in the general pneumatic conveying of coal, cement, rice, starch, etc. For intermittent work, such as pneumatic cash and mail conveying, its prompt response to overloads allows the use of a comparatively small driver. In coke-oven-gas manufacture the centrifugal compressor maintains constant suction on the gas main, and then compresses the gas so that it will flow through the condensers, purifiers, and into the gas holders. It is frequently used as a booster to a high-pressure reciprocating compressor, the two compressors forming together a very compact and efficient set. By compressing the air, say, to 30 lb. per square inch gage in the centrifugal compressor, the volume of the air to be handled by the reciprocating compressor is only about one-third of what it would otherwise be, making it possible to employ a much smaller unit.

While a centrifugal compressor will maintain a fairly constant pressure over a wide range of quantities, there is for every speed a certain range of quantity at which the discharge vanes cease to co-operate, causing a sudden drop in pressure. This "break-down" region can be pushed back toward lighter loads, and the drop in pressure made less abrupt by making the discharge vanes very few and their inlet angle small. Also, when working on that part of the pressure curve where the pressure increases with the quantity or remains constant, there are usually pressure surges or pulsations which, while slight in themselves, may be greatly intensified by a sort of resonance effect if the volume of the inlet and of the discharge piping happens to have a certain critical value. A slight throttling of the inlet will always stop these pulsations by making the pressure curve slightly drooping.

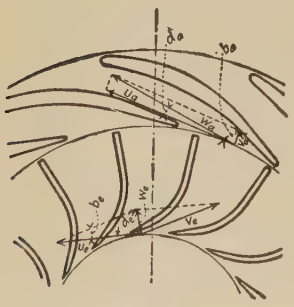


FIG. 34.

THEORY

Notation.—Referring to Fig. 34 let

$D_e(D_a)$ = impeller inlet (exit) diameter, feet.
$u_e(u_a)$ = impeller inlet (exit) peripheral velocity, feet per second.
$w_e(w_a)$ = absolute inlet (exit) velocity of gas, feet per second.
$v_e(v_a)$ = relative inlet (exit) velocity of gas, feet per second.
$b_e(b_a)$ = impeller inlet (exit) angle, degrees.
d_e = angle between w_e and u_e, degrees.
d_a = angle between w_a and u_a, degrees, or inlet angle of discharge vanes, if any.
p_1 = initial pressure of the gas, including the velocity energy (if any), pounds per square inch.
T_1 = temperature of gas corresponding to p_1, degrees Fahrenheit absolute.
d_1 = density of gas corresponding to p_1, pounds per cubic foot.

p_2, T_2, d_2 = corresponding values for the final conditions of the gas as it leaves the compressor.

γ = ratio of specific heats of gas, taken as 1.41.

$A = (T_2/T_1) - 1 = (p_2/p_1)^{(\gamma-1)K} - 1 = (p_2/p_1)^{0.289} - 1$.

s = specific gravity of gas at inlet conditions, referred to that of "free air" as unity.

I = quantity of inlet gas, cubic feet per second.

e_h = hydraulic efficiency, referred to adiabatic compression.

H = the theoretical (or total) head, or height against which the gas is raised, feet, including all hydraulic losses.

N = revolutions per minute.

Total Pressure Rise.—The fundamental equation giving the value of H is the same as that for H_b (see "Centrifugal Pumps," p. 115). For single-stage compression,

$$A = \left(\frac{p_2}{p_1}\right)^{(\gamma-1)/\gamma} - 1 = \frac{e_h d_1(\gamma - 1)H}{144 p_1 \gamma} = \frac{e_h d_1(\gamma - 1)u_a^2}{4{,}631 p_1 \gamma}$$

when $d_e = 90$ deg. (no inlet guide vanes) and $b_a = 90$ deg. (radial discharge). Assuming $p_1 = 14.7$, $T_1 = 520$, $\gamma = 1.41$, $e_h = 0.72$, and substituting, $p_2 = 14.7[1 + (u_a^2 s/4{,}300{,}000)]^{3.44}$. For small pressure rise, with $d_e = 90$ deg., $p_2 - p_1 = 0.0000165 e_h u_a^2 s[1 + (v_a/u_a) \cos b_a]$.

Fluid input horsepower is the horsepower applied to the gas and is independent of the actual pressure rise obtained. Fluid input horsepower $= Qd_1H/550$. For $d_e = 90$ deg., this becomes $0.00000432\, Qsu_a^2[1 + (v_a/u_a) \times \cos b_a]$.

Theoretical Horsepower.—This is the horsepower necessary to compress (and deliver) Q cu. ft. of gas per second from p_1 to p_2.

Theoretical horsepower $= 0.901Qp_1(p_2/p_1)^{0.286} - 1$; or, for small pressure rise $= 0.2618Qp_e$, where $p_e = p_2 - p_1$. This formula may also be used for higher pressures if $p_1 = 14.7$ lb. per square inch and p_e $(= 50.6A)$ is regarded as the mean effective pressure for adiabatic compression from p_1 to p_2. Table 13 gives values of the mean effective pressure p_e against values of $p_2 - p_1$ for the usual case of $p_1 = 14.7$.

TABLE 13

$(p_2 - 14.7)$, lb. per square inch	5	10	15	20	25	30	35	40	45	50
p_e, lb. per square inch	4.5	8.2	11.4	14.3	16.9	19.3	21.5	23.5	25.4	27.2

Table 14 gives the theoretical horsepower necessary to compress adiabatically (and deliver) 100 cu. ft. of air per minute from 14.7 lb. absolute to various gage pressures.

TABLE 14

Final gage pressure, lb. per square inch	5	10	15	20	25	30	35	40	45	50
Theoretical horsepower per 100 cu. ft.	1.95	3.58	4.99	6.25	7.37	8.39	9.35	10.23	11.06	11.84

Hydraulic Losses.—The hydraulic losses are the losses in pressure caused by the gas friction and by the sudden changes in the gas velocity or direction of flow. On the basis of D. W. Taylor's experiments on the flow of air in pipes, the pressure drop in the suction and in the discharge pipes (pounds per square inch) $= L = lv^2 s/400{,}000D$, where l is the length of pipe, feet, v the velocity of gas, feet per second, s the specific gravity of gas referred to free air (0.0764) as unity, and D the diameter of pipe, inches. For pipes of first-class workmanship and in very best condition, this loss may be reduced by about 20 per cent. The same care to have smooth pipe walls and to avoid too short bends should be taken with gases as with liquids.

Hydraulic Efficiency.—The hydraulic efficiency is the ratio of the theoretical horsepower to the fluid input horsepower, or $e_h = 184.8AT_1/H$. For $d_e = 90°$,

$$e_h = \frac{5{,}955AT_1}{u_a^2[1 + (v_a^2/u_a^2)\cos b_a]}. \quad \text{Also, } e_h = \frac{60{,}600(p_2 - p_1)}{su_a^2[1 + (v_a^2/u_a^2)\cos b_a]}$$

for small pressure rise, with $d_e = 90$ deg.

Other Losses.—The rotation loss, or friction loss of the impeller, considered as a flat disk of negligible axial width, as it rotates in the compressed gas, may be obtained from the formula: Rotation loss (horsepower) = $0.0737 \times (u_a/1{,}000)^3 D_a^2 d_m$, where d_m is the mean density of the gas (between p_1 and p_2), pounds per cubic foot. At full load this is practically the only loss to be considered, and it is this loss, more frequently than the centrifugal stresses, that determines the pressure rise that may be developed by a single impeller. Short-circuit losses result from the return of part of the gas through the impeller to the inlet instead of proceeding to the exit. Part takes place through the axial clearance between the impeller and the casing and decreases as the gas passed by the pump increases. Another part passes up the backs of the vanes, where the static pressure is lowest and the relative velocity highest, and at light loads, passes down the front of the vanes where the static pressure is highest and relative velocity highest. Short-circuit losses are lowest at full load. They are roughly proportional to the impeller outer diameter, the impeller exit width, and the passage height at the discharge-vane inlet. At no load the short-circuit loss is between two and four times the rotation loss, while at full load it is fairly negligible.

Centrifugal Compressor Constants and Characteristic Curves—Quantity Constant.—The quantity of gas delivered by a centrifugal compressor is proportional to $u_a D_a b_a$, or, quantity constant = $u_a D_a b_a$. *Compressor Constant.*—A compressor model can be used with practically the same efficiency for various combinations of quantity and pressure such that $K = Q/\sqrt{p_e}$, where K is the compressor constant, Q and p_e are the desired quantity and mean effective pressure, respectively. The compressor constant can be more conveniently written as $K = QN^2/p_1^{3/2}$.

Two compressors are similar when they have the same compressor constant. In similar compressors all impeller and discharge-vane linear dimensions are in the same ratio as their impeller diameters, while their impeller and discharge vane angles are respectively equal. For the same revolutions per minute the quantities delivered by similar compressors will vary as the cubes of their diameters; the pressures will vary as the squares of their diameters, and the shaft powers will vary as the fifth powers of their diameters. For the same wheel speed the quantity and the power will vary as the square of the diameter, while the pressure will remain constant.

Compressor Coefficients.—In order to make the tests on different compressors, or on the same compressor under different circumstances, comparable on a common basis, various coefficients are computed corresponding to the given observations and these coefficients are plotted as characteristic curves. From these curves the pressure, power, and the hydraulic and shaft efficiencies can readily be computed for any quantity of gas and any revolution per minute. The departure of the ratio v_a/u_a from that value for which the compressor was designed determines largely the efficiency of operation. Therefore, v_a/u_a, or its equivalent, Q/u_a, is generally used as the abscissa for the characteristic curves of a compressor, and it is designated as the load coefficient, C_e. It is also frequently represented by Q/N. The fluid input coefficient (C_i) represents the horsepower corresponding to any given observation divided by the cube of the wheel speed. For the case of axial impeller inlet and radial impeller exit, $C_i = 0.00000432Qs/u_a = 0.00000432C_e s$. Evidently the characteristic curve of C_i against C_e is a straight line making an angle with C_e whose tangent is $0.00000432s$; it may be drawn independently of the actual observations. In general, $C_i = 0.00000216s\ QV^2/u_a^3$, where $V^2 = 2gH = u_a^2 + w_a^2 - v_a^2 - u_e^2 - w_e^2 + v_e^2$.

The *pressure coefficient* corresponding to the observed pressure rise $(p_2 - p_1)$, is $C_p = AT_1/u_a^2$, (for notation see p. 176). The *theoretical power coefficient* is $C_t = 0.02571 C_e C_p$. The characteristic curve for C_t should be computed and drawn from readings from the *smooth* curve of C_p against C_e. The *shaft power coefficient* is C_s = shaft horsepower/u_a^3. The *rotation loss coefficient* is $C_r = 0.0737 \times 10^{-6} D_a d_m$, where d_m is the average density of the gas between p_1 and p_2, lb. per cubic foot. By adding the values of C_r to the values of C_i, a characteristic curve (practically a straight line) of fluid input plus rotation loss is obtained, and this curve will in a correctly designed compressor nearly touch the shaft power characteristic curve at the value of Q/u_a corresponding to the rated load of the compressor.

The ratio of C_t to C_i for any value of C_e gives the hydraulic efficiency e_h for that particular load. (For $C_e = 0$, the general formula $e_h = 5{,}955\ C_p$ must be used). Similarly, ratios of C_t to C_s give values of e_s, the shaft efficiency. The efficiency curves thus obtained will of course be smoother and more reliable than if the efficiencies were computed directly from the individual observations of pressure and power.

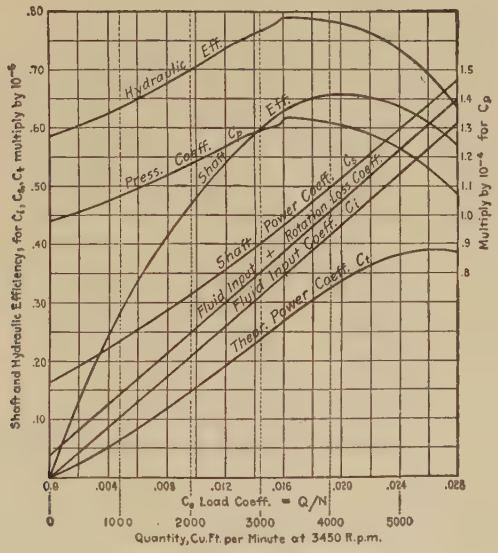

Shaft and Hydraulic Efficiency, for C_i, C_s, C_t multiply by 10^{-6}
Multiply by 10^{-4} for C_p
Hydraulic Eff.
Press. Coeff. C_p
Shaft Eff.
Shaft Power Coeff. C_s
Shaft Fluid Input + Rotation Loss Coeff. C_i
Fluid Input Coeff. C_i
Theor. Power Coeff. C_t
C_e Load Coeff. = Q/N
Quantity, Cu. Ft. per Minute at 3450 R.p.m.

FIG. 35.—Characteristic curves of centrifugal compressors.

Uses of Characteristic Curves.—Besides affording smooth curves of hydraulic and shaft efficiencies, a set of characteristic curves as shown in Fig. 35 enables one readily to draw reliable pressure and power curves against quantity for any given wheel speed or revolutions per minute. In that case, the load-coefficient scale may be replaced by a quantity scale, while the readings of the C_p and the C_s curves will give the data for the corresponding pressures and powers. (In Fig. 35 the wheel speed has, in all the coefficients, been replaced by the revolutions per minute). The system of characteristic curves is found to hold true for various sizes of centrifugal compressors, centrifugal blowers, and centrifugal pumps, giving in each case consistent curves regardless of the actual speeds, pressures and powers. Only in ventilating

fan blowers do there seem to be occasionally some serious discrepancies near the point of maximum efficiency.[1]

Multi-stage Centrifugal Compressors.—A multi-stage compressor consists of a number of single-stage compressors connected in series. When the number of single-stage compressors is small they are usually enclosed in the same casing; but when the number exceeds eight or ten they are generally subdivided between two or more casings, separated by intercoolers. Multi-stage compressors are frequently built with single-inlet impellers, depending for the overcoming of the axial thrust on balancing pistons, on special grouping of the impellers, and on similar devices; also with backward-discharge impellers, when they are supplied with substantial shrouds to prevent the blades from bending or breaking under the action of centrifugal force. They are also built with all the impellers of the double-inlet type, thus obviating the need of all balancing means and

[1] **Numerical Examples. Power Required.**—Find the power required to compress adiabatically 20,000 cu. ft. of air per minute from atmosphere to 30 lb. per square inch gage, the shaft efficiency of the compressor being 75 per cent.

Solution.—From Table 14, p. 177, the theoretical horsepower is $8.4 \times 20{,}000/100 = 1{,}680$, and the shaft horsepower is $1{,}680/0.75 = 2{,}240$.

Equivalent Suction Pressure.—What suction can be obtained with a compressor rated to deliver 2,500 cu. ft. of air per minute against 2 lb. per square inch gage?

Solution.—The compressor is rated for an *initial* pressure of 14.7 lb. per square inch. Since the *pressure ratio* depends only on the wheel speed, the hydraulic efficiency and the initial temperature, all of which are supposed to remain the same, the initial suction pressure is $(14.7/16.7) \times 14.7 = 12.94$ lb. per square inch absolute, and the suction obtained is $14.70 - 12.94 = 1.76$ lb. per square inch.

Equivalent Pressure when Compressing Gas.—What pressure can be obtained when compressing water gas with a standard unit rated 25,000 cu. ft. of air per minute and 15 lb. per square inch gage, and what power will be required if it requires 2,000 horsepower to compress the air to 15 lb. pressure?

Solution.—The density of water gas is 0.05167, and its specific gravity (compared with air) is 0.677. The mean effective pressure (m.e.p.) corresponding to 15 lb. per square inch is 11.44 for air (see also Table 1). For water gas, everything remaining the same, m.e.p. $= 11.44 \times 0.677 = 7.74$ lb. per square inch, and from Table 1 the corresponding final pressure is 9.3 lb. per square inch gage. The theoretical power for the air rating (from Table 2) is $5 \times 25{,}000/100 = 1{,}250$ horsepower. For water gas the theoretical horsepower is $1{,}250 \times 0.677 = 846$ hp., and the actual power $= 2{,}000 \times 0.677 = 1{,}354$ hp.

Equivalent Rating at Other Speeds.—A standard centrifugal compressor rated 4,500 cu. ft. of air and 15 lb. per square inch pressure is to be speeded up from 3,450 r.p.m. to 4,000 r.p.m. What increase of pressure and of quantity will result?

Solution.—The mean effective pressure for 15 lb. per square inch is 11.44. At 4,000 r.p.m. it will increase to $(4{,}000/3{,}450)^2 \times 11.44 = 15.38$, and the corresponding final pressure from Table 1 is 22.05 lb. per square inch absolute. Also the new quantity will be $(4{,}000/3{,}450) \times 4{,}500 = 5{,}220$ cubic feet per minute if the same ratio of v_a/v_a, and therefore the same hydraulic efficiency, is to be maintained (see "Quantity Constant" and "Load Coefficient," page 178).

General Problem.—What standard compressor can be used to exhaust 18,500 cu. ft. of anthracite producer gas per minute against a suction of 7 lb. per square inch? The compressor is to be installed 2,000 ft. above sea level. What horsepower is required?

Solution.—The barometer at 2,000 ft. altitude is 13.56. The compressor is therefore required to compress the gas from $13.56 - 7.00 = 6.56$ lb. per square inch absolute to 13.56 lb. This is equivalent to compressing the gas at sea level to a final pressure of $(13.56/6.56) \times 14.7 = 30.4$ lb. per square inch absolute or 15.7 lb. gage. The density of anthracite producer gas is 0.065 lb. per cubic feet. The mean effective pressure corresponding to 15.7 final pressure is (Table 1) 11.9 lb. per square inch. The corresponding mean effective pressure for air is $(0.0764/0.065) \times 11.9 = 14.0$, and the final pressure from Table 1 is 19.4 bl. per square inch gage. Suppose the nearest standard compressor is rated 16,000 cn. ft. per minute, and 15 lb. per square inch at 3,200 r.p.m. The mean effective pressure for 15 lb. is 11.44 and for 19.4 is 14.0. The standard compressor must therefore be speeded up to $3{,}200 \times \sqrt{14/11.44}$ or 3,540 r.p.m. For a constant "load coefficient" Q/N, the new quantity will be $16{,}000 \times 3{,}540/3{,}200 = 17{,}700$. So if it is desired to use the standard compressor and save the extra cost of a special size, then only 17,700 cu. ft. per minute of the gas can be exhausted with a suction of 7 lb. per square inch gage, or the compressor can be speeded up to handle 18,500 cu. ft. per minute, but the suction will be somewhat above 7 lb. If the horsepower required by the standard compressor is 1,300 (corresponding to a shaft efficiency of 0.6.15) the horsepower for the desired conditions will be $1{,}300 \times (17{,}700/16{,}000) \times (11.9/11.44) = 1{,}495$.

requiring a much smaller number of impellers; the interstage passages, however, become rather complicated.

The successive impellers in a multi-stage compressor, handling smaller and smaller volumes of air, should be designed on the principle of similar compressors if all stages are to be equally efficient. Since the revolutions per minute is the same for all impellers, the diameters and all the other dimensions should vary *inversely as the cube root of the density.* In practice, however, the impellers are divided into two or more groups, and each group designed for its own average conditions.

Cooling.—The cooling of the gas during its passage through a multi-stage compressor is of paramount importance for high efficiency and low power consumption, and ample passages for cooling water must be provided in the diaphragms between the stages. For pressures below 50 lb. per square inch, it is generally aimed to keep the temperature down to that corresponding to adiabatic compression; while for higher pressures, isothermal compression is more usually aimed at by the introduction of intercoolers between groups of stages. In the latter case, the hydraulic efficiency is given on the basis of isothermal compression.

For adiabatic compression, if the cooling is just enough to remove the heat above that corresponding to adiabatic compression:

$$p_2/p_1 = (1 + e_h d_1(\gamma - 1)u_a^2 S/144 p_1 g\gamma)^{\gamma/(\gamma - 1)}$$

where S is the number of stages, while the other symbols have the same meaning as on page 176. If the cooling is such that each stage starts with the same temperature, the temperature corresponding to p_1, then

$$p_2/p_1 = [1 + e_h d_1\ (\gamma - 1)u_a^2/144 p_1 g\gamma]^{S(\gamma - 1)/\gamma}$$

For strictly isothermal compression,

$$\log_{10}\frac{p_2}{p_1} = \frac{e_h d_1 u_a^2 S}{144 p_1 g \times 2.302} = \frac{e_h d_1 u_a^2 S}{10{,}663 p_1}$$

In all these formulas axial inlet flow and radial impeller exit are assumed; for all other cases u_a^2/g should be replaced by the proper value for H.

Theoretical Power.—The theoretical horsepower required to compress adiabatically and deliver 100 cu. ft. of gas per minute (initial pressure, p_1 lb. per square inch), is $\text{hp.}_a = 1.501 p_1[(p_2/p_1)^{0.29} - 1]$; and for isothermal compression it is: $\text{hp.}_i = 1.004 p_1 \log_{10}(p_2/p_1)$. Table 15 gives the ratio of the theoretical work for isothermal compression to that for adiabatic compression for the largest range of pressures likely to be met with in practice.

TABLE 15

p_2/p_1	1.5	2.0	2.5	3	4	5	6	7	8	9	10
(Isoth. ÷ Adiab.)	.940	.904	.875	.85	.812	.784	.763	.744	.728	.715	.703

Leakage.—To reduce the leakage between stages and from the inlet of the last stage to the atmosphere, labyrinth packings are usually provided. In general, the leakage may be assumed as about 3 per cent of the rated quantity, about two-thirds of this taking place between the inlet of the last stage and the atmosphere. This loss is fairly independent of the number of stages.

Centrifugal Compressor Tests.—In making acceptance tests, the compressor is tested with no piping at the inlet, the air being taken directly into the unrestricted compressor inlet, which is to be at a distance from the wall or from the floor equal to at least its own diameter. At the discharge end a length of pipe equal to about ten times the discharge diameter is attached. At about the middle

of this pipe, a blast gate is inserted, while at the far end the measuring orifice, or a header containing a number of small orifices, is bolted on.

The orifice is usually selected to pass a quantity about 35 per cent greater than the rated quantity. The following rough formulas are frequently used: $Q = 109d^2\sqrt{P} = 21d^2\sqrt{i}$, where Q = cubic feet of air per minute; d = diameter of orifice, inches; P = pressure above atmosphere, pounds per square inch, and i = inches of water. Readings are generally taken at about six or seven different loads, the readings taken being the inlet temperature, the discharge pressure, the orifice pressure and temperature and the revolutions per minute. The discharge pressure is taken with an impact tube inserted into the discharge pipe about midway between the compressor and the blast gate, pointing upstream and reaching a little more than half way into the pipe. A second impact tube nearer the blast gate is desirable. The orifice pressure is taken with an impact tube placed opposite the center of the orifice at a distance from the plane of the orifice equal to about one-fifth of the orifice diameter. The orifice temperature is taken in the straight part of the pipe before the orifice. The bare bulb of the thermometer should protrude for about 2 in. into the stream of air, the air velocity rarely exceeding 60 or 70 ft. per second.

The efficiency of a centrifugal compressor is the ratio of the theoretical power, adiabatic or isothermal, corresponding to the rated quantity and pressure, divided by the power delivered to the compressor shaft. There is, however, considerable room for controversy and misunderstanding in this connection, owing to the difficulty of measuring accurately the output of the driver. The input of the driver, however, is generally easily determinable. It is therefore customary to specify that the input of the driver per 100 cu. ft. of air delivered by the compressor shall not exceed so many kilowatt-hours, or so many pounds of steam, the steam conditions being those of the customer's plant.

Commercial sizes of the single-stage compressors have pressure ratings from 0.75 to 4 lb. per square inch and capacities from a lower limit of 500 cu. ft. per minute to a higher limit which ranges from 12,000 cu. ft. at 0.75 lb. pressure down to 3,000 cu. ft. at 4 lb. pressure. The multi-stage compressors are built of the following sizes:

Cubic feet per minute	4,500	9,000	16,000	25,000	40,000	50,000
Pressure, pounds per square inch	6–35	6–25	6–25	12–30	12–30	12–30

Multi-stage compressors have also been built for pressures as high as 150 lb. per square inch, the volumes being comparatively small.

With a driver allowing speed variation, like a steam turbine or direct-current motor a centrifugal compressor of a given rating will operate with a number of different combinations of pressure and volume without excessive loss in efficiency. The different rated pressures of multi-stage compressors are usually obtained by the use of a different number of stages, all stages in a given size of machine being similar. A given compressor may, however, operate successfully between, say, 15 and 30 lb. per square inch, if the driver will allow of the necessary speed variation.

Without any special governing device a centrifugal compressor driven at constant speed will maintain a substantially constant pressure, regardless of the volume of air delivered. Should there be temporarily no demand for air while the compressor is being driven at full speed, the casing will become slightly hotter, but otherwise the machine will not be subjected to any particular strains. The power required by a centrifugal compressor between half-load and, say, 25 per cent overload varies approximately as the load. The following table shows the usual variation of pressure and power with the volume handled by a single-stage compressor at constant speed.

For blast-furnace purposes, where a constant quantity of air is to be delivered against varying pressures, a constant-volume governor has been developed.

CHARACTERISTICS OF CENTRIFUGAL COMPRESSORS AT CONSTANT SPEED

Volume, per cent of full load	0	20	40	50	60	70	80	100	120
Pressure, per cent of full load	92	94	97.5	100	103	105	105	100	91
Power, per cent of full load	50	53	58	62	66	73	81	100	120

BLOWERS AND COMPRESSORS

Rotary blowers are built for air pressures varying from 6 oz. to 10 lb. or even 12 lb. per square inch. The best efficiencies of this type of blower, however are usually secured below 5 lb. pressure, but the simplicity of the machine gives it an advantage over compressors of the piston type and frequently warrants its installation for the higher pressures indicated when designed for this purpose. As the machine operates by displacement, it is usually preferred for cupola practice, and copper and lead blast furnaces, because its positive action will not permit a reduction in air supply if the cupola tends to clog. For other uses of air at pressures below 8 oz. the fan is ordinarily more economical.

Blowers of this type may be arranged to give either constant volume or constant pressure, and to handle either liquids or gases. They consist of a casing containing one or more revolving impellers of various forms of design. Figure 36 represents a cross-section of the Sturtevant high-pressure blower, which is built in capacities ranging from 5 to 15,000 cu. ft. per minute at 8 oz. pressure (speeds, from 375 to 800 r.p.m. for the smallest size down to 160 to 220 r.p.m. for the largest; weights with sub-base range from 200 to 39,000 lb.). The smaller machines have a vertical arrangement of shafts, while the larger types usually have their shafts in the same horizontal plane. Two impeller blades are always in action, and leakage by one is caught by the other. The proper size of blower for a cupola may be calculated on the basis of 30,000 cu. ft. of air per ton of iron melted. Sturtevant machines are also used or handling gases. The capacities for the various sizes of gas exhausters range from 7,500 to 900,000 cu. ft. per hour at 8 oz. pressure, making no allowance for shrinkage, which will vary from 10 to 20 per cent, depending on the gas and its pressure. The inlet and outlet diameters run from 3 in. in the smallest size to 30 in. in the largest; weights, from 400 to 33,000 lb.

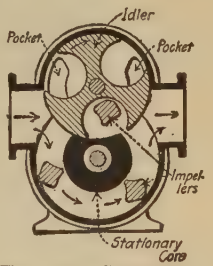


FIG. 36.—Sturtevant rotary blower.

Figure 37 illustrates a cross-section of a Roots blower. The two impellers are symmetrical and are driven in opposite directions by gears outside the casing. The impellers do not touch each other nor the casing, but the clearance is reduced to a minimum in order to reduce slip or leakage. The amount of this slip or leakage may be determined by operating the machine with a closed discharge, at a speed sufficient to maintain the required discharge pressure. The amount is usually largest in machines of smallest capacity, *i.e.*, a machine displacing 0.75 cu. ft. per revolution at a pressure of 1 lb. will have a slip of from 60 to 70 revolutions, while a machine having a capacity of 300 cu. ft. per revolution will have a slip of from 3 to 5 revolutions. For intermediate capacities the slip will vary proportionally and increase with higher pressures as the square root of the discharge pressure, *i.e.*, at 4 lb. pressure the slip will be approximately twice that at 1 lb.

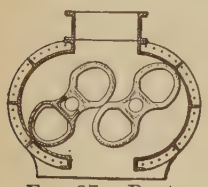


FIG. 37.—Roots blower.

In most blower work the so-called hydraulic formula for horsepower will be found satisfactory: horsepower $= Q(p_2 - p_1)/33{,}000$, where Q is the cubic feet of

air compressed per minute, p_1 the initial pressure and p_2 the final pressure, pounds per square foot. To get the actual horsepower at the shaft, the horsepower should be divided by the efficiency, which will vary from 0.60 to 0.80. If the speed remains unchanged, owing to the small clearances, the horsepower tends to increase tremendously with decrease in orifice.

The so called Hytor compressor (or exhaust) Fig. 38, works on the principle of a number of columns of liquid acting acting as pistons between the rotor teeth. The casing is ovoid and as the moving liquid swings away from the rotor shaft the gas is

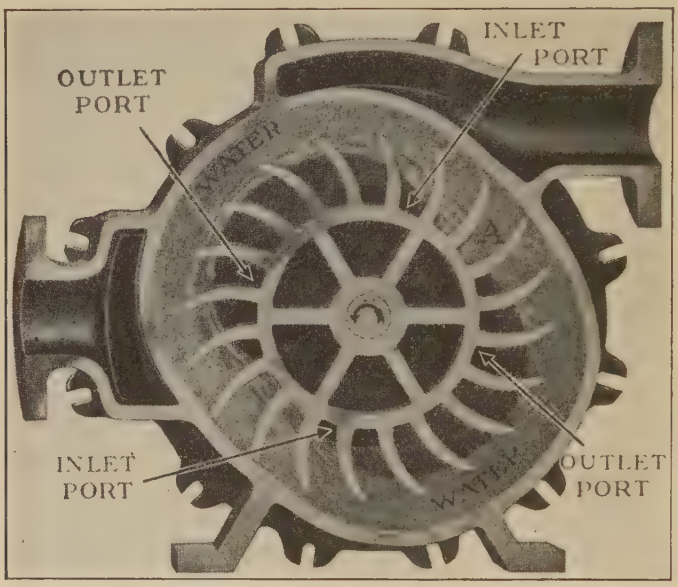


FIG. 38.—Hytor compressor.

drawn into the space vacated. As the liquid swings back further in the revolution the gas is compressed and tends to escape through the outlets.

The efficiency is high but the effluent gas will always be saturated with the vapor of the liquid used as the circulating medium.

CHIMNEYS

Chimney draft is based upon the difference of specific gravity of cold air and heated air. The column of warm air in the chimney exerts a pressure per square foot at the base of hd_1, where h is the height in feet and d_1 is the density of hot gases in pounds per cubic foot. The pressure of the outside air for the same height is hd_2, where d_2 is the density of cold air in pounds per cubic foot. The motive force is therefore the difference of these two, or $h(d_2 - d_1)$, in pounds per square foot.

For ordinary calculations,

$$D_1 = H\left(\frac{7.64}{T_2} - \frac{7.95}{T_1}\right)$$

Where H = height of chimney in feet, D_1 = intensity of draft, inches of water, T_1 = absolute Fahrenheit temperature of chimney gases, T_2 = absolute Fahrenheit temperature of outside air, and P_2 = observed atmospheric pressure, pounds per square inch. For high altitudes above sea level:

$$D_1 = 0.52HP_2\left(\frac{1}{T_2} - \frac{1}{T_1}\right)$$

This gives the maximum theoretical draft with no flow taking place. The actual draft obtained is about 20 per cent less, due to chimney friction, and head required to create velocity of gases.

Kent's draught formula is

$$H = \left(\frac{0.3\text{hp.}}{E}\right)^2$$

$$E = A - 0.6\sqrt{A}$$

Where E = effective area of chimney, A = actual area (cross-section) of chimney, H = height at top in feet above grates and horsepower = total connected boiler horsepower. Based on a maximum combustion of 5 lb. coal per rated boiler horsepower per hour, this formula may be modified to

$$H = \left(\frac{0.06W}{E}\right)^2$$

where W = total pounds coal fired per hour under connected boilers. This form is more suitable for high rating conditions. See p. 31 *et seq.*

Mechanical Draft.—From the preceding discussion, it is evident that for high rates of combustion the stack becomes impracticably high or the sensible heat loss due to high flue temperature becomes too large for economy. To mitigate this, mechanical or artificial draft of some form may be employed.

The theoretical pressure produced by a revolving fan wheel is given by Murgue as

$$H = \frac{U^2}{g}$$

where H = maximum pressure difference, between fan suction and discharge, in feet of air; U = velocity of fan blade tips, feet per second; and g = 32.2, acceleration due to gravity. Air pressure in inches of water column is generally referred to in blast and draft.

$$h = \frac{SH}{144p}$$

where h = inches of water pressure; S = weight of 1 cu. ft. air at 75°F. (usual room temperature), or 0.074495 lb.; and p = pressure of 1 in. water column in pound per square inch, or 0.3061 lb.

$$U = \frac{2\pi rn}{60}$$

where r = radius to tip of blade in inches and n = r.p.m. Hence

$$h = \left(\frac{2\pi rn}{60}\right)^2 S = C(rn)^2 S$$

The draft loss through a B. & W. boiler with standard baffling, clean, varies from 0.10 in. or 0.15 in. at rating, to 0.29 in. to 0.45 in. at 200 per cent rating and 0.54 in. to 0.65 in. at 300 per cent rating. The loss in flues of fairly uniform cross-section is given as 0.1 in. per 100 ft. straight flue, and 0.05 in. for 90-deg. bends. But in the irregular flue shapes often necessary, double these values are not infrequent.[1] The draught loss through economizers varies from 0.25 in. to 0.70 in. The sum of all these losses gives the required chimney draft. The diameter is computed from the weight of coal burned. With stokers using blast it is necessary only to allow 0.1 in. draft over the fire at maximum capacity, as the only function of the stack is to draw off the gases without pressure in the furnace, the resistance of fire bed and stoker being overcome by the blast pressure.

Brick Chimneys.—The red-brick chimney cannot be less than 0.7 ft. thick at the thinnest portion; in other respects it is designed for stability against a wind pressure of 50 lb. per square foot and against crushing. The walls, therefore, are

[1] H. C. Meyer, Jr. "Steam Power Plants," McGraw-Hill Book Co., 1912.

always thicker at the bottom, and the stack is tapered. This type is not often used for power stations. Radial-brick chimneys may be 0.58 ft. thick at the thinnest part, as the brick is specially molded; the cross-section is always circular,

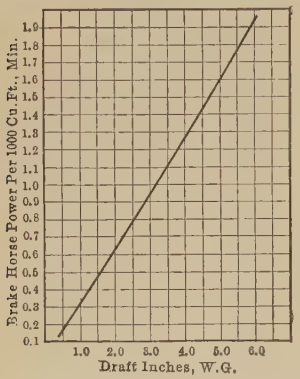


Fig. 39.—Required horsepower for forced-draft fans.

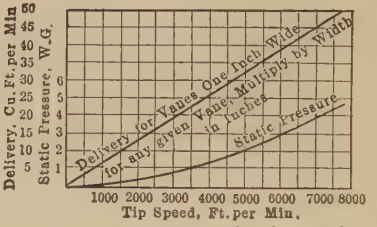


Fig. 40.—Power and volume of forced-draft fans.

and tapers in the same manner as red-brick chimneys. Brick chimneys are the most durable of all the types. The firebrick portions of lining need be carried up only 30 ft. above the grates.

Steel chimneys have the advantages of lightness and strength, but since they are better conductors of heat, must be lined with brick for heights over 75 ft. except in forced-draft installations. They must be carefully inspected and painted from time to time, as they are subject to deterioration by corrosion.

Reinforced concrete is much stronger than brick and will stand high tensile strains like the steel chimney. The stack is therefore often built straight like the steel chimney, and is always considerably lighter than brick, as it may safely be much thinner. It is usually poured in 5-ft. or 6-ft. sections, which may be carried up a section a day, making erection rapid. No lining is required other than the short section of firebrick above the grates (30 ft.). It is one of the cheapest and most durable forms if well designed and built, but like all reinforced concrete, is dependent upon care and watchfulness during construction.

Foundations for brick chimneys are now made almost exclusively of concrete, and are designed on the basis of proper bearing values, like any other foundation. They are usually spread or stepped out at the foot, in order to provide sufficient resistance to overturning from wind pressure.

Cost of Brick Stacks

Approximate boiler horsepower	Height	Diameter	Diameter square base outside	Price
85	80	2 ft. 1 in.	7 ft. 5 in.	598.00
135	90	2 ft. 6 in.	8 ft. 3 in.	786.00
200	100	2 ft. 11 in.	9 ft. 10 in.	1,226.00
300	110	3 ft. 7 in.	10 ft. 2 in.	1,492.00
450	120	4 ft. 3 in.	11 ft. 2 in.	1,785.00

Brick stacks may be figured on a basis of $12 per thousand for laying with masons at $0.55 per hour.[1]

The cost of concrete stacks is about 5 to 10 per cent less than brick.

REFERENCES: Gustav Herrmann, "Die Graphische Theorie der Turbinen und Kreiselpumpen." Julius Ritter von Hauer, "Ventilationsmaschinen der Bergwerke" and "Die Wettermaschinen." Louis Ser, "Traité de Physique Industriale," pp. 668–723. J. Boulvin, "Cours de Mécanique Appliqué aux Machines." Bryan Donkin, "Experiments on Centrifugal Fans," *Proc. Inst. C. E.*, vol. cxxii, part 4, 1895. Charles H. Innes, "The Fan." Simons, "Compressed Air," McGraw-Hill. Peele, "Compressed Air Plants," Wiley. Harris, "Compressed Air Theory and Computation," McGraw-Hill. Innes, "Air Compressors and Blowing Engines," Van Nostrand. Thorkelson, "Air Compression and Transmission," McGraw-Hill. Von Ihering, "Die Gebläse," Springer, Berlin. Hirsch, "Die Luftpumpen," Jänecke, Hanover. Ostertag, "Kolben und Turbo-compressoren," Springer, Berlin. Zerkowitz, "Thermodynamik der Turbomaschinen," Oldenbourg, Berlin.

COMPRESSION, STORAGE AND TRANSPORTATION OF GASES

BY A. CRESSY MORRISON[2]

The problem of compressing gases is a complex one because of the great differences between them. Gases vary far more widely among themselves in their physical and chemical characteristics than do either liquids or solids.

There is 50 per cent more difference between the relative specific gravities of carbon dioxide and hydrogen than there is between the weight of water and of mercury, or of lead and white pine. Nitrogen is an inert, passive, dead substance. Fluorine is one of the most energetic substances in nature. Hydrogen, nitrogen and chlorine are chemical elements. Acetylene, hydrogen chloride and methane are combinations of two or more elements.

Oxygen is the supporter of both animal and vegetable life. Cyanogen is the most actively poisonous substance known. Fire or burning is a combination of fuel with oxygen. Carbon dioxide is used as a fire extinguisher. Nitrogen will not burn. Hydrogen burns with a colorless flame and acetylene possesses 240 candlepower. A gas explosion is a combination or association of oxygen with a fuel. A gunpowder or a dynamite explosion is a dissociation of nitrogen from the other elements in the explosive.

Hydrogen chloride is an exceedingly strong acid. Ammonia is an almost equally strong alkali. Methane is neutral. Carbon dioxide has a negative heat value of 14,500 B.t.u. for each pound of contained carbon. Hydrogen has a positive heat value of 62,100 B.t.u. per pound, being greater than that of any other known substance.

Carbon dioxide is so heavy that it can be left for a time in an open bucket, or beaker, or can be siphoned or poured like water from one vessel to another. Hydrogen is the lightest of all substances—is used for balloons—and at the same time in chemical action is just as distinctly a metal as is zinc, copper, iron or silver.

Oxygen is the most abundant substance in the universe. Krypton and xenon are so rare as almost to leave doubts as to their existence. Methane is practically insoluble in water. Ammonia is absorbed to the extent of 1,000 times the volume of water. Chlorine may be liquefied and solidified with very meager pressures, and at atmospheric temperature. It is doubtful if hydrogen or methane have ever been even liquified, certainly not without the most extreme conditions of pressure and refrigeration.

[1] Further cost data may be found in Gebhart's "Power Plants," Wiley, 1912.

[2] Secretary of International Acelytene Assn.

Air, hydrogen and methane are odorless, colorless, and tasteless and may be swallowed or inhaled with impunity. Chlorine has a strong, pungent, unbreathable odor, is greenish in color, and not only cannot be breathed or swallowed, but produces severe chemical burns, if brought in contact with the skin.

Laws Governing Behavior of Gases.—From the above, it would be apparent that no two gases have exactly similar characteristics, and that no general rules can be laid down governing the handling of all gases in actual practice. There are, however, a number of laws governing the behavior of true gases under theoretical conditions, briefly as follows:

1. Gases exert equal pressures on all parts of their containers and in all directions, and at all points within their mass.
2. The volume of any gas varies inversely with the pressure, the temperature remaining constant (see p. 144).
3. The volume of a gas, the pressure being constant, is proportional to its absolute temperature.
4. The density of elementary gases is proportional to their atomic weights. The density of compound gases is proportional to one-half their molecular weights, hydrogen being one in both cases.
5. A vapor not near its saturation point, that is, out of contact with its liquid, behaves in all respects like a true gas.
6. The density of a saturated gas cannot be increased by increase in pressure or decrease of temperature. Liquefaction will occur in consequence of such increase or decrease, which will prevent increase in pressure.
7. The only gas which can prevent a liquid from evaporating is its own vapor.
8. Mixtures of all gases exert pressure against the containing vessel equal to the sum of the individul pressures exerted by th e different component parts of the mixture. The pressure exerted by any single gas is equal to the sum of the pressures of the different portions of the volume of that gas; that is, one-third of the gas in a container will exert one-third of the total pressure if that third is in the container alone.
9. The flow of gases through an orifice is in inverse proportion to the square-root of the density of the gas; that is, hydrogen with a density of one, will escape four times as fast as oxygen with a density of 16, through the same orifice under similar conditions.

The above are the ordinary laws governing the behavior of gases. There are, however, variations from these rules. Theoretically the pressure-volume, or pv curve of any gas should be an equilateral hyperbola. As a matter of fact, as partially discussed on page 144, the pv curve of a number of gases varies slightly from an equilateral hyperbola. Methane at 300 lb. shows a variation of as much as 4 per cent; other gases vary under low pressure and not at high, and *vice versa*. True pv curves can be plotted only from experiments; those for most of the industrial gases have been worked out by the various industries handling the gases.

The volume of a gas varies directly with changes in its temperature referred to absolute zero (−492°F.), pressure remaining constant, and conversely the pressure varies directly as the temperature, the volume remaining constant. Roughly speaking, a variation of 5° (4.92°) makes a difference of 1 per cent in the volume, or in the pressure of the gas.

Gases are Transported, Stored and Used, either Compressed, Liquefied or Absorbed.—It may be said that any gas can be liquefied if all conditions of pressure and refrigeration are sufficient, and all gases are soluble to some extent in

some solvent. The physical and chemical characteristics of each gas taken in the light of the limits of industrial conditions determine in what shape each gas is to be stored and used.

Gases having a very high liquefaction pressure, and not being soluble to a great extent in any other substance, are sold compressed; among these are oxygen, nitrogen, hydrogen, coal-gas, air, and most of the other so-called true gases.

Gases which liquefy within industrial limits of pressure and at atmospheric temperature are shipped as liquids, among which are chlorine, ammonia, blaugas, nitrous oxide, carbon dioxide, etc. Gases which are soluble to a large extent in some economical solvent, are sold in solution; among these are ammonia, hydrochloric acid and acetylene.

It is necessary to compress all of these gases whatever shape they are to be sold in. Gases are compressed by a combination of pressure and cooling. Jacket water is cheaper than power; hence, the best industrial compressors are of multi-stage construction, those for high pressures having as many as five stages with intercoolers between each stage.

Most gases when in a liquid state expand faster for an increase in temperature than do the steel containers in which they are to be shipped, and hence care must be taken to leave sufficient space for expansion above the liquid in the containers. When gases are to be handled either liquefied or absorbed, the containers holding them have to be filled by weight instead of by pressure. As noted above, an increase of pressure or decrease in volume of a saturated gas will not increase the gage reading, but will result in an accumulation of liquefied or dissolved gas. The pressure will remain the same, but the weight in the container will increase.

The gases which are sold in solution are absorbed in water for the most part. Acetylene might be sold either compressed or liquefied as the liquefaction pressure is approximately 700 lb. at atmospheric temperature. Acetylene, however, acquires explosive properties when compressed in excess of some 20 lb., and hence is not handled unabsorbed in either liquefied or compressed form. Acetylene is extremely soluble in and loses its explosive properties when absorbed in acetone, which is in turn absorbed in asbestos or some other porous non-inflammable material.[1]

In the handling of gases, the lubrication of the compressors is a question which requires consideration. The compressors handling the inert gases can be lubricated with oil. The compressors handling oxygen must not be lubricated with oil; but must use uncompounded graphite or a solution of soap in water, or glycerine in water. Chlorine in the presence of water has a very corrosive effect on iron; and water cannot be used with chlorine in contact with iron. Chlorine has a destructive action upon oil. In the compression of this gas, concentrated sulphuric acid is used as a lubricant.

In handling compressed gas of any sort avoid any severe shocks to the drums; avoid any mixture within the drums or in any confined space of two gases that may react with one another; avoid any sudden temperature changes in the neighborhood of the drums; and if the gas is expanding through the gage-cock in the drum, keep it warm enough to avoid freezing the effluent gas. Avoid using any lubricants in the compressors that could, if volatilized, react with the gas being compressed. If the gas being compressed is combustible, avoid air leaks in the suction end. After all, a compressed gas is a high explosive and should be handled with the respect due the genus.

[1] Certain "garage experts" have been advising a mixture of acetylene and oxygen in a single cylinder as being more effective than the gases mixed in the burner. If this advice is followed to any considerable extent the coroners may anticipate some busy times. The mixture is itself likely to detonate, even were there no danger of the flame striking back through the burner.—EDITOR.

Van der Waals' Gas Constants

	a	b
Air	0.00257	0.001560
Ammonia	0.00798	0.002992
Carbon monoxide	0.00275	0.001683
Carbon dioxide	0.00717	0.001908
Chlorine	0.01110	0.002154
Ethane	0.01074	0.002848
Ethylene	0.00886	0.002533
Hydrogen	0.00042	0.000880
Hydrochloric acid	0.00695	0.001728
Hydrogen sulphide	0.00888	0 001926
Methane	0.00366	0.001591
Nitrogen	0.00259	0.001650
Nitric oxide (NO)	0.00257	0.001160
Oxygen	0.00273	0.001420
Sulphur dioxide	0.01316	0.002486
Nitrogen monoxide (N_2O)	0.00720	0.001888

SECTION V

CRUSHING AND GRINDING

By H. A. Megraw[1]

The size reduction of materials of various kinds forms an important step in the technical operation of many industries, so that, in the course of time, a particular system of machinery has been designed to perform the work simply and economically. The principles of division and subdivision are essentially the same for all solids, whether they may be rocks, minerals or composites. Reduction machinery, however, has been designed particularly for operating upon friable or non-elastic materials. Substances that are highly elastic or plastic, or which become plastic under small temperature rises, are not suitable for subdivision in machines of the ordinary type. They are to be handled by machinery coming under the classification of cutting, rending or abrasion machines. Such materials as rubber, wood, gums, rosins and some chemical compounds are not to be broken or pulverized successfully in the usual reduction machines, but must be turned over to machinery of a different type.

At the same time there are materials that lie on the border line between friable and elastic substances that can be reduced in size by appliances embodying the principles of standard crushing equipment, though somewhat modified for this special service. Thus many materials can be ground satisfactorily in the old-fashioned buhr-stone mill which could not be handled so well with more modern machinery. The buhr mill employs roughened hard stone faces moving one upon the other, the effect being to rend or tear the material to be ground, as well as to abrade it.

Buhr stones would not be considered satisfactory appliances for ordinary breaking or pulverizing, due to the high rate of wear and the consequent expense of maintenance, as well as small capacity, but for many special problems they provide the only solution. Among these are many compounds beside the exceptional ones above listed.

Industrial Rock Reduction.—There are many industries in which the breaking, grinding and pulverizing of rock, stone or ore is an essential feature. Among these industries may be mentioned quarrying, cement-making, mining, glass-making, lime manufacture, pottery, sand and gravel production and many others. They involve the reduction in size of rock from one point to another, the character or size of the product depending entirely upon the service for which it is intended. Rock reduction alone in the United States amounts to many millions of tons yearly for all purposes. In view of the extent of this operation, it is somewhat surprising that more attention is not given to the principles, mechanics and costs of such work.

The several stages of rock-breaking may be divided generally into three, which for the purpose of this paper may be called **breaking, crushing,** and **grinding.**

Rock-breaking may be limited to the primary operation of reducing by machinery

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the rock from the size at which it comes from the mine or quarry, to a more or less uniform size suitable for secondary reduction. This secondary reduction, whether it be accomplished in one or more stages, may be called crushing. Reduction to finely granular or pulverulent condition may be placed within the class of grinding. These classes are purely arbitrary, but are used as a convenience to distinguish between the departments of rock reduction for which ordinarily machines of different type and character are used.

General Principles.—There are several mechanical means by which rock is commercially reduced from one size to another. These include compression, roll action, beam action, impact and attrition. Practically all machines use one of these principles or a combination of two or more of them.

Compression has been used for a great many years. It involves the compression or squeezing of rock between two rigid surfaces. One or both of the surfaces may be movable, the rock being fractured by direct compression between two planes. Machines embodying this principle are ordinarily not the most efficient, since they are necessarily reciprocating and require direct application of power without any material leverage.

Roll action is the process of reducing rock by means of a hard-faced solid wheel traveling over the rock and breaking it by crushing between the wheel and a rigid surface. While it is a form of compression, it is not the same as the direct-compression method and its results are different. Roll action may be accomplished by means of a wheel traveling over a stationary surface or by means of passing the rock between two wheels wherein the reduction is accomplished. This is a feature of many well-known machines.

Beam action accomplishes rock reduction by applying power at an unsupported part of the rock between two rigidly fixed points. This method is a feature of certain crushing machines, which will be mentioned later, and is perhaps one of the most efficient and economical methods of doing this work.

Impact is the crushing of rock by means of dropping a weight upon it. Impact crushing may embody features of compression and beam action, but does not involve roll action. It is one of the most efficient means of breaking rock and has been used for years in certain forms and is still used in a modified way.

Attrition accomplishes rock reduction by abrasion or by rubbing the rock between two surfaces, one of which may be fixed and the other moving, or both of which may be moving in the same direction but at a different rate of speed, or in different directions. Attrition is one of the most expensive and inconvenient methods of rock reduction, since it reduces the grinding medium to an excessive extent. Attrition is admissible where the rock may be made to grind upon itself, but in practice this has been accomplished only in an extremely limited way.

Some machines accomplish the reduction by the employment of only one of these methods, but the majority of them combine two or more of them.

Angle of Nip.—In most crushing machines, the two faces between which the crushing is performed are not parallel, but are inclined to each other. Obviously, this angle, known as the angle of nip, must be sharp enough to prevent any slipping of the rock engaged between the crushing surfaces. If the angle is too obtuse, the rock will slip, or may even be pushed out from between the crushing faces. There is, consequently, a continual effort at compromise between the endeavor to produce an efficient machine, in which the nip angle must be sharp enough to prevent slipping, and the commercial effort to produce a machine having as large an opening as possible for a fixed weight—necessarily involving a more obtuse angle of nip.

Angle of nip in its broader sense may be said to be the angle formed between the planes of crushing surfaces acting upon the particle to be crushed. For example, in the case of gravity stamps, the crushing surfaces are parallel and angle of nip is zero and need not be considered. On the other hand, in breakers and rolls the crushing surfaces are not parallel and the ore particles to be crushed are held between two inclined surfaces approaching one another and exerting pressure upon the ore particle.

To those engaged in rock or ore crushing, there are two conditions of angle of nip which are of special interest. The first may be termed "the critical angle of nip" and the second may be called the "safe" or "practical" angle of nip. The critical angle of nip may be defined as that angle formed by the crushing surfaces, at which the ore particle is just on the dividing line between being held between the crushing surfaces and being shot out from between them by pressure. Or, in more scientific language, it is that angle where the components of the forces, including friction, acting to hold the particle in between the crushing surfaces, are just balanced by the components of the forces tending to snap the particle outward. "The safe or proper angle of nip" is a somewhat less angle than the "critical angle" and represents the angle between the crushing surfaces at which there is practically no possibility that the ore particle can be snapped out. This angle is a variable depending upon various factors but, for general average conditions, in the case of crushing rolls, Richards ("Ore Dressing," Vol. III, p. 1226) has estimated that 32 deg. represents the best figure to be taken. Since rolls have curved surfaces, the only way that angle of nip can be figured in their case is to take that angle formed by the intersection of the two tangents drawn between the two points of contact of the ore particles against the rolls.[1]

In the case of breakers, both of the jaw and gyratory types, the angle of nip must also be considered and it is to be presumed that the general angle of 32 deg. applies equally well. The factors which alter the angle, that is, the safe working angle, are the character of the ore, the character of the crushing surfaces, the speed of approach of the crushing surfaces and perhaps, in some cases, also the way that the rock is fed. The character of the ore affects it because a smooth, hard particle or a soft, greasy particle will have a lower coefficient of friction than an average gritty particle of rock and will, therefore, require a less angle of nip than 32 deg. as the safe angle. Similarly, crushing surfaces which have become worn smooth and are of a hard, glassy nature will have a lower coefficient of friction than the average crushing surfaces and will require reduction of angle of nip. The effect of speed is perhaps less than that of the two preceding factors, although it will be found in general that where crushing surfaces are moving toward one another very rapidly, the coefficient of friction is, in effect, reduced by this high speed and thus a lowering of the angle nip brought about. This is very clearly brought out at times by crushing rolls where ore particles will not be nipped while the rolls are running, but will frequently become nipped after the rolls have been disconnected from power and just before they come to a stop. The effect of manner of feeding may have also a minor effect. If rock is fed dry, the coefficient of friction is higher than it is if the material is fed with a little moisture in it. On the other hand, if the rock is fed with a stream of water, then this stream of water may wash away lubricating clay, etc., and have a tendency to increase the coefficient of friction. Speed of feeding likewise may have minor effect in that an ore particle fed upon swiftly moving crushing surfaces is not immediately accelerated to the speed of that surface and hence may fail to be nipped. This is especially true in the case of crushing rolls and means that the high-speed rolls usually require the use of an angle of nip somewhat less than 32 deg., in order to be safe.

Rock-breaking Machinery.—In the field of primary rock breaking there are

[1] See RICHARDS, "Ore Dressing," Vol. III, p. 1226, and Vol. IV, p. 89.

two types of machines that are largely, almost exclusively, used. These are the jaw type and the gyratory type. Of the jaw type of machine, there are two modifications, one the Blake type and the other the Dodge. In general, **Jaw Breakers** consist of a heavy iron or steel frame embodying one fixed jaw and one movable jaw. The movable jaw is suspended from the shaft or hinge forming the

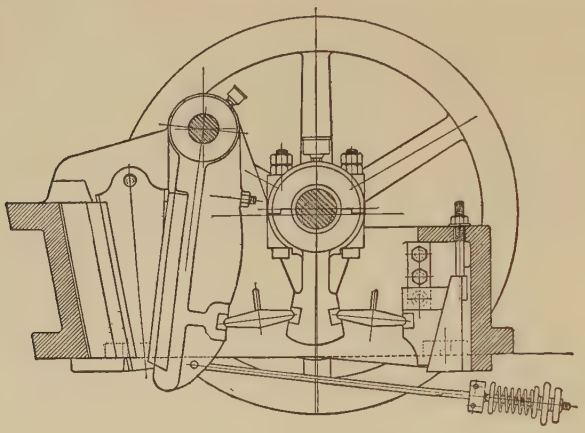


FIG. 1.—Blake crusher.

upper end of the jaw, the reciprocating movement being applied at or near its lower end. The movement is applied by means of a cam on the driven shaft of the machine, which raises and lowers a pitman, which imparts motion through

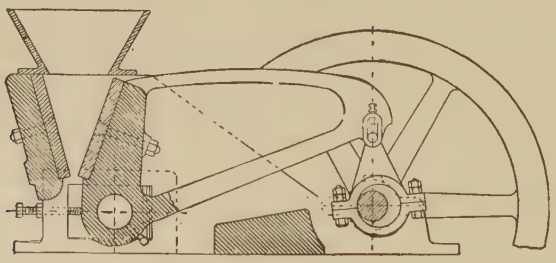


FIG. 2.—Dodge crusher.

a pair of toggles. As the cam raises the pitman, the toggles assume an angular position, the swinging jaw being held against the front toggle by means of a rod and coil spring, as shown in the accompanying illustration. During the succeeding semi-revolution of the shaft, the cam lowers the pitman, straightening out the toggles, thus forcing forward the lower end of the swinging jaw. The rock between the two jaws of the machine is thereby compressed and ruptured. The wear is taken upon removable jaw plates, which are fitted to both the stationary and moving jaw. The features of this machine include a rectangular opening which will take a large rock in comparison to the total weight of the machine. When the swinging jaw recedes from the stationary one, the discharge opening is increased in size, with the result that the discharge varies in size between wide limits. The product of machines of this type may consist of slabs of rock nearly

as thick as the maximum opening of the swinging jaw, as wide as the width of the rectangular opening and of indefinite length.

As the swinging jaw closes, the angle of nip becomes greater and the tendency is more and more to push the rock out of the machine, rather than to crush it. This is particularly noticeable on rocks of a slippery nature and in any machine where the limit of the angle of nip has been closely approximated by the manufacturer, which is nearly always the case.

Breakers of this type are accomplishing useful work during only half the time they are in operation, the other half being occupied in retreating the moving jaw and getting ready for another compressing operation. Consequently, the proportion of power applied to overcoming friction is high. The product of such machines is not desirable if uniform material is required and its use is only permissible when a machine of a different type is used for secondary crushing, or under special conditions where

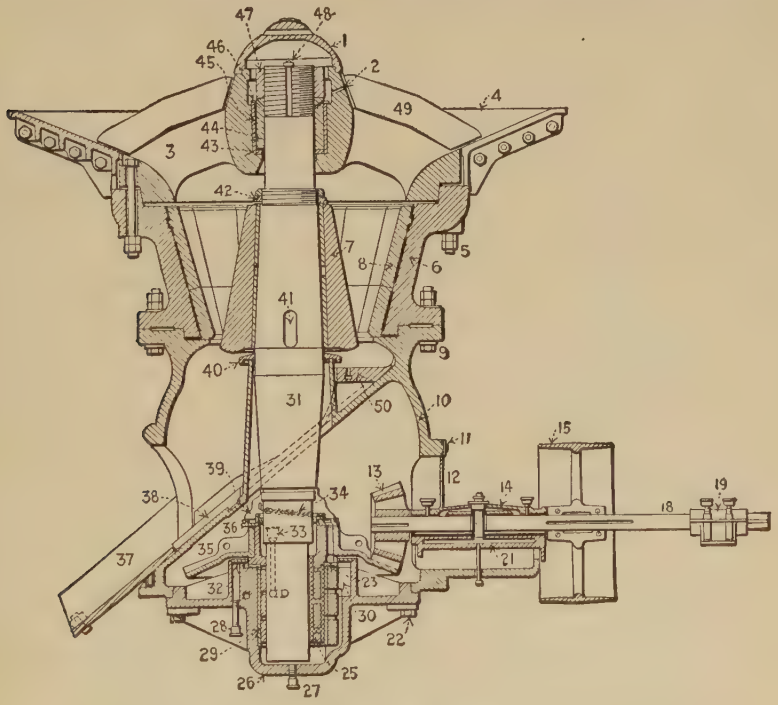


FIG. 3.—Gyratory breaker.

uniformity of product is not required. These machines are widely used in spite of their manifest disadvantages because of their comparatively large capacity and low installation cost.

The Dodge type of jaw breaker embodies the same principles as those of the Blake type. The essential difference between the two is that in the Dodge type the movable jaw is hinged at the bottom instead of the top. Its mechanics are illustrated in the accompanying illustration. The result of using a movable jaw, hinged at the bottom, is that the discharge opening is much more nearly constant so that a product more uniform in size character is obtained. It still maintains the disadvantage of delivering a slabby or tabular product. The closing of the movable jaw lessens the angle of nip, thus tending to wedge the rock more tightly between the jaws and to accom-

COMPARISON OF GYRATORY AND BLAKE TYPE JAW BREAKERS

	Gyratory	Jaw	Gyratory	Jaw	Gyratory	Jaw	Gyratory	Jaw	Gyratory	Jaw	Gyratory	Jaw
Crusher opening	7 by 52	by 10	15 by 112	15 by 30	30 by 230	30 by 48	36 by 264	36 by 48	48 by 360	48 by 60	60 by 432	60 by 96
Size of product	2½	72½	2½	4	5	6	6	6	7	7	9	10
Weight of crusher, pounds	10,000	6,000	70,000	39,000	225,000	80,000	400,000	95,000	500,000	140,000	700,000	500,000
Capacity, tons per hour	15	8	100	48	750	95	1,000	125	1,200	165	1,350	600
Horsepower	10	7	53	30	150	110	175	150	218	175	275	275
Capacity, tons ÷ weight, tons	3	2.66	2.86	2.5	6.7	2.37	5	2.42	4.8	2.35	3.8	2.4
Capacity, tons ÷ horsepower	1.5	1.14	1.89	1.6	5	0.863	5.71	0.833	5.50	0.94	4.9	2.18

plish more effective breaking. Dodge type machines are not widely used, due to their comparatively small capacity and consequently, increased installation cost per ton of product.

Gyratory.—The second type of rock machine ordinarily used is the **Gyratory.** Essentially this type of breaker consists of a central pillar shaft fixed at its uppermost point and provided with means of gyration at its lowest point. On this shaft is fixed a renewable crushing head in the shape of a frustum of an upright cone. This head gyrates inside of an inverted conical concave, the inner surface being lined with renewable wearing plates. The receiving opening of machines of this type is, therefore, annular and breaking is accomplished by the alternate advance and recession of the crushing head with respect to the concaves. Since the movement is gyratory, it is continuous, so that while the head is approaching one point of the concave, it is receding from the point diametrically opposite. There are no reciprocating elements and the power is applied continuously to crushing work. Since the point of greatest gyration is usually some distance below the point of breaking, the advantage of leverage is embodied.

In this machine, the breaking is effected by a combination of compression, roll action and beam action. As the crushing surfaces approach each other, compression is brought into play and since the crushing head is always traveling, the friction between the rock and the head causes it to revolve, thereby bringing into play roll action since the head rolls over the rock being broken.

Since in an annular opening of this kind, a tabular piece of rock would necessarily be fixed at its ends and unsupported at its middle point, beam action is brought into play and the rock will be broken at about its middle point. Since the radius of gyration is not great in any case, the discharge opening does not vary greatly and the angle of nip is not subject to great variation. As a result, the product of machines of this type is much more uniform than with other types, and more cubical in character.

The gyratory breaker is practically a stage-breaking machine, since a piece of rock approximating the maximum capacity of the angular opening will be reduced in successive stages by means of successive nips of the large piece and of the resulting smaller pieces into

which the rock is broken. Due to the small gyrating movement, each nip accomplishes only a small part of the total work, a number of successive nips being required to break the rock from its original size to the finer product of the machine.

Gyratory breakers are much more efficient than those of the jaw type with regard to tonnage produced, whether on a basis of horsepower consumed or total weight of the machine. The accompanying table shows the comparison of Blake jaw type and gyratory breakers based on equal width of opening in each case. It will be seen that the superiority of the gyratory over the jaw breaker increases with the size of the machine up to a certain limit.

There has been considerable argument over the comparative advantages of the jaw and gyratory type of rock breakers; one of the principal advantages claimed by the advocates of the former being that less head room is required for its use. This apparent advantage has been found to be fallacious in general. By superimposing the drawings of jaw and gyratory breakers of equal opening width, it will be found that there is little if any difference between the two as regards the difference of elevation between the point of intake and the point of rock discharge.

There are, however, cases in which it is desirable to use breakers of the jaw type. In general, these are where a limited capacity of rock of certain size is required. For example, should it be required to produce 200 tons per 24 hr. of rock of maximum initial size of 15 in., it will be seen that the jaw breaker has a capacity of about 48 tons per hour, whereas the gyratory has a capacity of about 100 tons per hour. Since the gyratory machine is much heavier, it will be more expensive to install. Consequently, the jaw-type machine, although having a great excess over the required capacity, would be advisable because its capacity is not so enormously excessive as in the case of the gyratory, and is a lighter and cheaper machine to install. In many cases where rocks of large size are to be handled, even this advantage disappears, since it may often be much more economical to operate a large gyratory breaking machine for a small fraction of the total time than continuously to expend power, which is excessive per unit, by means of the jaw-type machine. Where continuous operation at maximum capacity is expected, or where uniformity of product is desirable, the gyratory machine is far preferable.

In both jaw and gyratory types of rock breakers, mechanical efficiency should be considered, as with any machine for any purpose. Rock breakers are not all alike and some of them have advantages which ought to be carefully considered in the selection of machines for any particular work.

There has been a great deal of discussion and difference of opinion with regard to the proper selection of a rock breaker for primary breaking. Rock coming from a mine or quarry is made up of pieces of various sizes ranging from fine granular pieces up to 5 or 6 ft. in diameter. In general, it may be said that the average piece will not be more than 24 to 36 in. in diameter. Some authorities advocate the installation of a primary breaker capable of handling the largest piece, while others advocate a breaker only large enough to handle the average pieces, breaking up extremely large ones with dynamite before taking them to the machine. The advantage of having a breaker of large size is that it avoids the expense of dynamiting or sledging with its consequent large employment of labor. Its disadvantage is that when a breaker is large enough to take the pieces of extreme size, there is a great excess of capacity, requiring the machine to be idle a greater part of the time. The large machine will, in addition, represent a greater investment than the smaller one and will require more power while it is actually in operation. In cases, which are by no means unusual, where a large breaker may be operated at definite periods when other equipment is not requiring power, thus avoiding a high power peak, it may be ultimately economical to use a large breaker, doing away with the expense of dynamiting and

sledging. This point is subject to investigation in each case, and is one that must be decided upon the merits of the specific conditions obtaining.

Power Requirements.—The size of gyratory crushers is usually indicated by an arbitrary number while jaw crushers are designated by inch measurements of the rectangular feed opening.

The table given herewith shows the approximate power requirements of various sizes of gyratory breakers. The power figures given should be used with discretion because of the factors which affect power requirements; the principal of these being the hardness of the rock being broken and the method of feeding. On soft rock, minimum power is required and on hard rock the power requirements increase rapidly. It is ordinarily true that breakers are supplied with more power than they require, that is, motors or engines of excessive power are supplied. The various makes of breakers will also have a definite effect upon power requirements so that with those best designed to obviate high friction, a minimum of power would be required. The starting torque of a gyratory breaker is small unless it should be stopped when full of rock. In such a case, it is usually necessary to dig it out before attempting to start the machine. The same conditions will obtain with jaw breakers.

AVERAGE POWER REQUIREMENTS OF GYRATORY BREAKERS

Size of opening on each side of spider, inches	Maximum cube that will enter breaker, inches	Power required, horsepower
7 by 28	7	10 to 20
8 by 34	8	15 to 25
10 by 40	10	20 to 35
12 by 44	12	25 to 40
15 by 55	15	50 to 70
18 by 68	18	65 to 100
21 by 76	21	100 to 140
24 by 99	24	125 to 175
36 by 132	36	150 to 200
42 by 162	42	175 to 200
48 by 180	48	200 to 225
54 by 198	54	200 to 250
60 by 216	60	250 to 300

The most favorable operating condition for jaw breakers is when the ratio of size of feed to that of product does not exceed 6:1. When this ratio is exceeded, the efficiency of the machine is greatly reduced, friction is increased and heating almost inevitable; the capacity is reduced to a fraction of normal, and in general the value of the machine is seriously impaired. The amount of material broken will be in proportion to the width of the discharge opening. That is to say that when reducing rock from 18 to 3 in., the machine will handle about three times as much material as when reducing from 6 to 1 in., all other factors remaining the same. The speed of the breaker directly affects its capacity.

The power required depends on the friction that has to be overcome in addition to the actual work of breaking. For the Blake type of breaker, there is one theoretically correct speed based on the law of gravity. The time of stroke of the movable jaw must not exceed the rate of movement of the material due to gravity, but there are mechanical limitations which will reduce the possible speed. An approximation of power required for the operation of Blake breakers is given in the accompanying table.

Approximate Power Required for Blake Jaw Breakers

Size	Horsepower	Size	Horsepower
10 by 7	7	48 by 36	150
12 by 9	10	48 by 42	150
16 by 10	12	54 by 42	150
24 by 10	14	60 by 42	160
28 by 13	25	60 by 48	175
30 by 15	30	60 by 60	200
36 by 18	40	84 by 60	225
42 by 24	50	90 by 60	250
48 by 26	60	96 by 60	275
48 by 30	110		

Secondary Crushing.—After the work of the primary breakers has been done, the rock is in pieces varying according to the size of the original breaker, from 2 or 2½ up to 6 in. in diameter. In industrial use, the rock, unless it be limestone flux or ore for furnace use, requires further reduction; therefore a second stage in reduction is required—this stage we classify as crushing. It may be performed in machines of a number of different kinds.

Ordinarily, crushing is done in machines of the same type as those used for primary breaking, but of smaller size and delivering a smaller product. Jaw crushers may be and are used quite often, but are not particularly well adapted for this kind of work because of the variable character of their products. Gyratory breakers of small size are often used and these are much more satisfactory, since they deliver a product more uniform in size, more cubical in character and have much greater capacity than crushers of the Blake type. Small gyratory crushers, operating at high speed, are available for this purpose and are very largely used. The characteristics of the jaw and gyratory types have already been described.

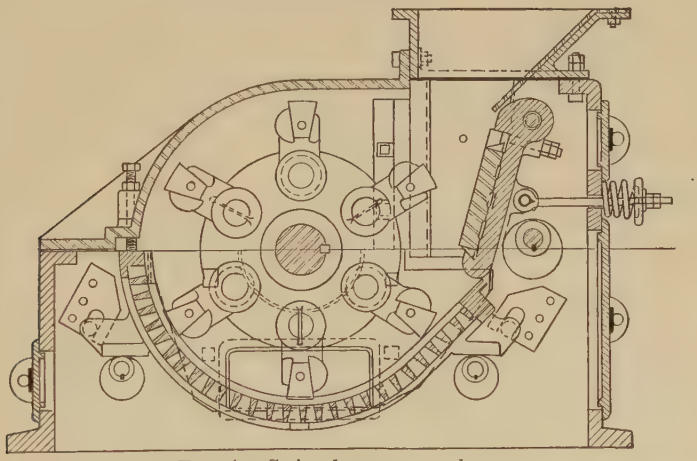


Fig. 4.—Swing-hammer crusher.

In addition to the gyratory and jaw types for crushing, there are others that are very largely used, among these may be mentioned the swing-hammer type, the roll type and other specially designed machines.

The Swing-hammer type of machine ordinarily consists of a central shaft to which are connected a number of hammers so pivoted as to swing freely. In operation, this central shaft revolves at high speed, the hammers swinging straight out through the effect of centrifugal force. The rock to be broken is fed over a breaking plate so set that the swinging hammers just miss touching it in their revolution. The hammers strike the rock and the impact of this blow while the rock is resting upon the breaker plate, breaks it into pieces. Since

the hammers swing freely, there is no danger of breaking the hammer by this operation. The under part of the machine, concentric with the shaft, consists of a grid of steel bars, the distance between them being equal to the desired product. If the blow of the hammers on the rock does not divide it small enough to pass through these apertures, the hammers continue to give the rock additional blows while it is on the grid, the eventual result being that all of it passes through these apertures.

Machines of this type are satisfactory for use with comparatively soft rocks such as shales, limestones, clays, etc., or on rock which does not contain an excessively high percentage of silica. These machines will take pieces up to 4, 5, or 6 in. in diameter and the economical fine limit is about ¼ in. It is practical to secure ¼ in. product on very soft material, but by diminishing the output size, the capacity of the machine is seriously reduced also. Under proper operating conditions and in fields to which the machine is adaptable, efficient results are secured. The power required is not high, but depends largely upon the character of the rock broken and the ratio of reduction. The accompanying drawing shows the general make-up of a machine of this type.

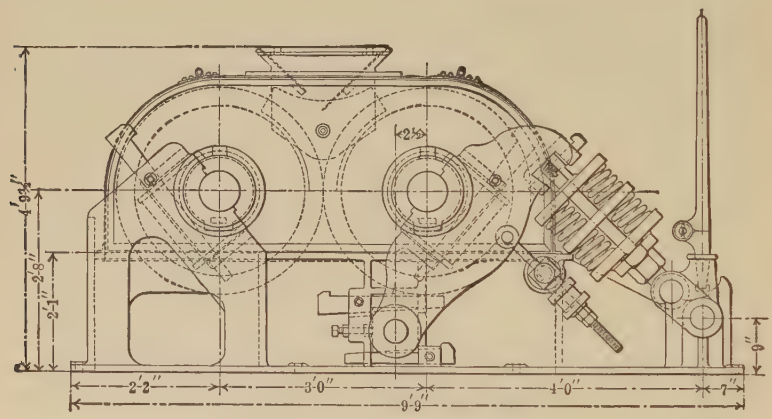


Fig. 5.—Crushing rolls.

The crushing roll is a type of machine that has been in use for a great many years. It consists of two solid cylinders with bearings in the same horizontal plane, set fairly close together, the stream of rock to be crushed passing between the two. One or both of these cylinders may be positively driven. They are usually adjustable so that the rock may be reduced to a desired size. Rolls are usually made with a solid cylindrical core having removable shells which take up wear. These shells are made of hard steel, often so alloyed as to present a surface highly resistive to abrasion.

The most favorable condition for crushing rolls is when the ratio of size of feed product does not exceed about 4 or 4½ : 1. When this ratio is exceeded, the efficiency of the roll is greatly decreased and the power requirement increased. With the ratio of 4 : 1, the power is practically constant whether the ratio is applied to large feed and large product, or fine feed and finer product. The capacity of the machine, however, will usually be in proportion to the size, that is, with production on large pieces, the capacity will be greater than with reduction on smaller pieces; other factors remaining constant.

The speed of course, is a very important item and ordinarily this should not exceed 900 ft. per minute, although there are rolls in successful operation at a speed of 1,000 ft. The difficulty is that with higher speeds, there is brought into play a grinding effect, due to slip of the rock particles on the roll shells, and this tends to abrade the shells and wear them out very rapidly. The best results are obtained by the use of moderate speed not exceeding 900 ft. per minute and with careful attention to the angle of nip, which has already been explained. The cylinder diameter of the rolls should always be such that the angle of nip will be sharp enough to grasp the piece presented and nip it without permitting slipping.

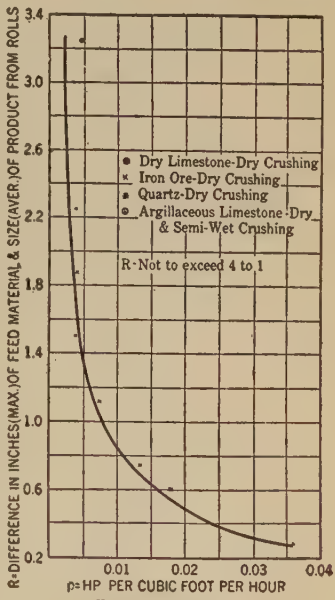
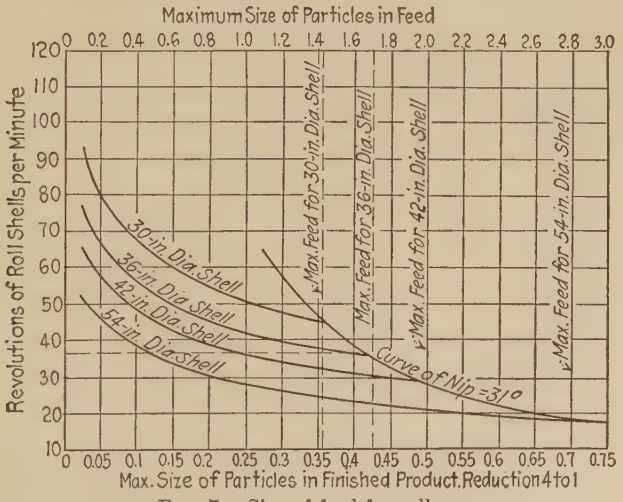


Fig. 6.—Power consumed in rolls.

One of the great difficulties with rolls has always been the impossibility of feeding them equally throughout the entire width of the roll surface. The result is that grooves are worn in the roll shells in certain places, which seriously diminish their efficiency and result in the production of a material that is not properly sized. Many devices have been tried to obviate this difficulty. Often one of the rolls is flanged, the opposite roll fitting into this flange, the idea being to permit definite distribution of the feed over the entire roll face without spilling any of it. This variation has not been altogether successful. A second method has been to provide one of the rolls with a longitudinal movement along its axis; this movement is known as "fleeting." Usually the fleeting motion is small, the idea being to present a new surface continuously at any point where the rock may be delivered, thus avoiding


Maximum Size of Particles in Feed
0 0.2 0.4 0.6 0.8 1.0 1.2 1.4 1.6 1.8 2.0 2.2 2.4 2.6 2.8 3.0
Revolutions of Roll Shells per Minute
120 110 100 90 80 70 60 50 40 30 20 10
30-in. Dia. Shell
36-in. Dia. Shell
42-in. Dia. Shell
54-in. Dia. Shell
Max. Feed for 30-in. Dia. Shell
Max. Feed for 36-in. Dia. Shell
Max. Feed for 42-in. Dia. Shell
Max. Feed for 54-in. Dia. Shell
Curve of Nip = 31°
0 0.05 0.1 0.15 0.2 0.25 0.3 0.35 0.4 0.45 0.5 0.55 0.6 0.65 0.7 0.75
Max. Size of Particles in Finished Product. Reduction 4 to 1
Fig. 7.—Size of feed for rolls.

doing all the work in one spot. Fleeting devices may be continuous and mechanical, or may be adjustable by hand. The fleeting device has been successful in a number

of instances, but it is the general conclusion that unequal wear cannot be entirely avoided by any means so far discovered. The roll is highly efficient under some conditions, particularly if properly designed as to the relation of the roll diameter to the size of feed, if operated at proper speed and given proper care. With abrasive feed, however, it is almost impossible to prevent unequal wear and other machines are often considered more advisable for such service.

The theoretical capacity of rolls may easily be calculated by computing the cubical contents of the ribbon passing through the machine, the data necessary includes the speed of the roll, the width of the roll face and the distance apart of the two cylinders. This will give the cubical volume that will pass over the rolls in a given time. In practice, due to the unequal character of the feed, not more than one-quarter of this cubical volume should be taken as the actual capacity of the machine.

As to power, the accompanying formulæ show the requirements for crushing rolls of all kinds. The power to drive rolls at normal speed, without load, varies for different sizes, speeds and makes of machines. An approximate idea being obtained from the following formula.[1]

$$P_1 = 0.0835(D + W)$$

When,

D = Diameter of rolls in inches, and
W = Face of rolls in inches.

The total power required to drive rolls, loaded is:

$$P = (p \times M) + P_1$$

When,

P = Total power,
p = Horsepower per cubic foot per hour (from Fig. 6, p. 201),
P_1 = Horsepower required to drive rolls without load, and
M = Capacity of rolls in cubic feet per hour.

FEED SIZE FOR 32-DEG. NIP ANGLE ON ROLLS

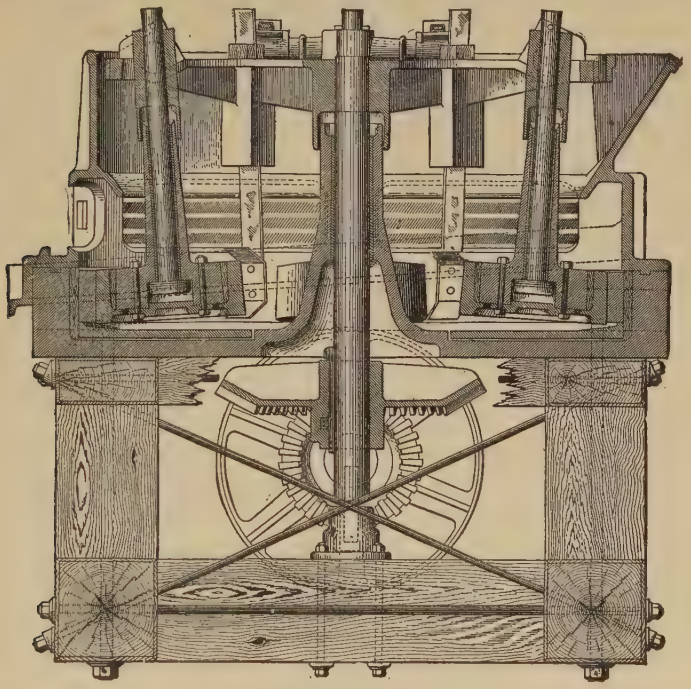
Roll, diameter, inches	Space between rolls, inches						
	¾	⅝	½	⅜	¼	⅛	0
	Maximum size of feed, inches						
36	2.23	2.10	1.96	1.84	1.71	1.57	1.45
30	1.99	1.86	1.73	1.60	1.47	1.34	1.21
26	1.83	1.70	1.56	1.44	1.31	1.17	1.05
24	1.74	1.61	1.48	1.36	1.22	1.10	0.96
20	1.58	1.46	1.32	1.20	1.06	0.94	0.80
16	1.42	1.29	1.16	1.03	0.90	0.77	0.64
9	1.14	1.01	0.88	0.75	0.62	0.49	0.36

In addition to the types already mentioned for crushing, there are used in various instances other devices such as, edge runners of the chilean mill type, ball mills and various specially designed machines for accomplishing this work.

The chilean mill is a well-known rock reduction machine, but since its field is usually in fine reduction, it will be considered more exhaustively under the head of grinding. For crushing purposes, chilean mills with comparatively slow move-

[1] These formulas from "Electric Motors in the Cement Industry" by R. B. WILLIAMSON, *Am. Inst. Elec. Eng.*, compiled from data gathered by the Committee on Industrial and Domestic Power.

ment and very heavy wheels have been used with success, but only at the finest stage which might be considered in the classification of "crushing." Material 1 to 2 in. in diameter may be fed to such machines, and a product varying from



(Sectional View.)

FIG. 8.—Huntington mill.

¼ in. to 20 mesh can be obtained from them. Since the most useful field for the chilean mill is in granulation or grinding, it will not be further considered here.

Ball mills are very widely used for reducing coarse rock to a product which varies with the ideas of the metallurgist installing the plant. Ball mills are in use

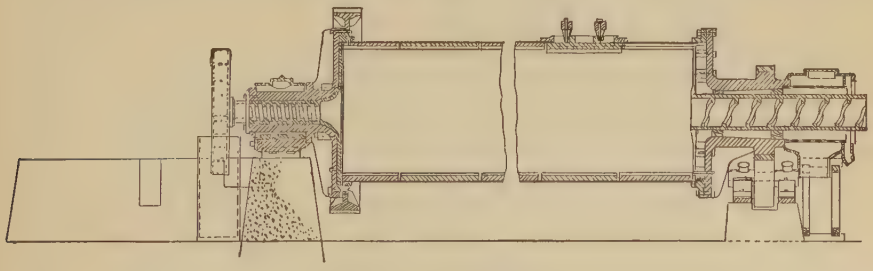


FIG. 9.—Tube mill.

to 10 ft. in diameter, and for crushing work are of short length, not over 5 or 6 ft. Rock is fed to these machines up to 3 in. in diameter, although such practice is seldom considered economical. The best field for ball-mill crushing is with feed at about 1½ in. or less with a product varying from ¼ in. to 20 or 30 mesh. In

many instances the coarse rock is received by a large diameter ball mill and reduced in one stage to 50 mesh or finer. Operation of mills in this way practically necessitates the use of a classifier or screen directly following the mill in which the material already sufficient ground may be removed from the pulp, and the unground material returned to the ball mill for further grinding. The capacity of such mills is quite large, but since ball and tube mills are usually considered as fine-reduction machines rather than crushers, this too will be considered under the head of grinding.

In addition to the ones already mentioned, there are a large number of machines that have been designed to accomplish this particular work. Some of them have been quite successful, and others not so much so. It would be impossible to consider the merits or demerits of each or any of them in a discussion of this kind, and consequently they are left for study in the light of the particular circumstances under which their use is contemplated. It may not be out of place, however, to consider here some of the principles upon which these machines are constructed.

Some crushing machines use in connection with compression a grinding or abrading motion. For example, crushers of the jaw type are in use in which the swinging jaw has, in addition to its swinging motion, a rolling action. These machines are advantageous in that they assist in the discharge of the material crushed. They should be used with discrimination, however, since with highly abrasive rock the wearing plates of such machines are subject to rapid deterioration. They are advisable for use with rock which is not abrasive, such as limestones, shales, clays, etc., which may be expected to have no cutting action. With quartz, granite or the various trap rocks and other hard, siliceous, crystalline aggregates they should not be used.

In general, any crushing machine which takes all of the crushing strain directly upon the bearing is not a long lived one. The high stresses developed momentarily and intermittently by the crushing of hard rock are very severe and, mechanically speaking, should not be delivered directly upon a bearing, which ordinarily cannot withstand such heavy work for any great length of time. The best-known types of machines are so constructed that the principal strains are not directly on the bearings, and these types should be used, particularly where very hard rocks are to be operated upon. With soft rocks which do not involve high-breaking strains, machines of this type may be used, but conditions should be carefully studied before installing them for any particular work.

Grinding.—The third stage in rock reduction, which we may call grinding, takes a feed from the secondary crushing machines and makes product which will vary according to the subsequent requirements. The desired product may be a granular aggregate varying in size from 10 to 200 mesh, or it may be an entirely −200-mesh material. For metallurgical purposes, such as cyanide treatment, flotation, etc., a very fine product is often desirable. For concentration, a granular product is required, while for various industrial purposes, such as glass making, pottery, sand production, etc., the requirements may be widely varied.

The machinery used for this stage will include many well-known types, such as grinding pans, edge runners, ring-roll mills, fine-grinding rolls and ball and tube mills. These machines involve one or a combination of two or more of all of the crushing principles that have already been enumerated. Their efficiency varies greatly, and durability or wearing qualities are just as variable. Their application will depend largely on the kind of rock to be handled and the use to which the product is to be put. These machines will be considered here according to the various types mentioned.

Grinding pans constitute one of the oldest types of fine-reduction machinery. There are many different types, but probably the best known of these is the Wheeler pan, which was formerly very widely used in amalgamating processes in silver mills. These machines comprise a pan or tub 3, 4 or 5 ft. in diameter, made of steel or wood, with a comparatively shallow depth, ordinarily approximating 3 or 4 ft. The bottom of the pan is equipped with hard-steel or iron plates upon which a muller operated through a central shaft grinds the ore. The muller is equipped with grinding shoes, adjustable so that its distance from the bottom may be any required measurement. The muller revolves, rubbing its shoes against the hard iron bottom of the pan. The grinding is performed by the abrasion of the material between these two hard surfaces.

In operation machines of this type produce very finely divided material, but since they involve rapid and costly destruction of the grinding surfaces, they are not considered economical for present-day use. Their capacity is not great, an ordinary 5-ft. pan being capable of handling 10 to 15 tons per day when reducing from 10 or 15 mesh to an average of 60- to 80-mesh material. The power required is from 10 to 20 hp., depending upon the character of the rock and fineness of the product.

Edge runners may be typified by the mill known as the chilean mill. There are many types of this machine, the usual type having straight grinding faces, but there are some variations having their grinding faces curved, one convex and the other concave. The chilean mill consists essentially of a bottom ring die over which rolls a heavy wheel, actuated through a central shaft. The type is an old one, having been originally used in crude form by the early Spanish miners in Spanish America. For comparatively coarse work the, machine is made with a rolling diameter comparatively large, reaching 6 or 7 ft. This machine is operated at comparatively slow speeds, usually 12 to 18 r.p.m., or sometimes even less. For fine reduction, the modern type of chilean mills has a running diameter of about 4 or 5 ft. and is operated at speeds up to 30 or 35 r.p.m.; in this case the wheels are smaller and lighter than with the large diameter mill. Chilean mills are operated either wet or dry. Their work is performed largely by rolling action, but one variation of the mill adds somewhat to its capacity by providing the factor of abrasion. This is by swinging the roller to what is essentially a crank. The horizontal shaft connecting with the driving mechanism of the center of the mill, is provided with an offset of 2 to 4 in., the shaft of the roller being connected to the offset. This gives to the rolling action a sliding or grinding effect so that the mill is dragged over the die in addition to rolling over it.

This variation is known as the Mantey offset. It increases the fineness of the product, but introduces the factor of abrasion which increases the wear on the grinding surface and consequently the cost of operation.

The power required for operating chilean mills is about five- to seven-tenths of a ton per horsepower-hour when producing a fine product. The capacity varies according to the work done. On comparatively coarse grinding, with slow-speed mills, in the neighborhood of 20 to 25 tons per hour may be reduced, while with high speeds this capacity is increased.

Ring-roll grinding machines are of the type in which the grinding anvil is a horizontal ring. The mullers swung from a spider attached to a central shaft, press against this vertical grinding surface. The Huntington mill is the best

known milling machine of this type. Ring-roll mills are high-speed devices and while they have been used for a number of years for all classes of rock reduction, their employment is now limited to dry or wet grinding of comparatively soft materials, such as coal, limestone, mineral earth, etc. The rapid wear of these machines under conditions of hard grinding militates against their use and their high speed makes them a rather difficult machine to control. They are, however, appropriate and economical for use in grinding coal for producing pulverized-coal fuel. They are widely used for this purpose and are said to be satisfactory.

In the case of grinding coal, the removal of the ground material is by an air current which may be regulated to take away the material sufficiently ground and to leave behind the particles which require further grinding. The capacity of a 50- or 54-in. diameter mill for this work will amount to about 1 ton per hour for each muller roll; the machines are usually made either with three, four or five rolls. The power for operation under these conditions varies according to the size of the machine and the excellence of the design, from 25 to 85 hp.

There are variations of the ring-roll type in which the ring is set vertically with rollers pressing against the inner horizontal face of the vertical ring and held there by spring pressure. These variations may have advantages, but such features must be considered with respect to each problem.

Crushing rolls, such as have already been described, are often used for the production of granular rock. A product as fine as 10 or even 20 mesh can be produced by these machines, but for such use they are generally considered uneconomical. Their very small capacity together with their tendency to groove and corrugate, delivering a product of widely varying sizes, makes them rather a high-cost and inefficient machine for such work.

The tube mill or ball tube mill is the machine generally considered most economical for producing either a finely ground product or a finely granular product. The tube mill consists essentially of a tube of varying diameter and length, installed horizontally and rotated about its horizontal axis. Ordinarily the shell is made of plate steel and the heads of cast iron or cast steel. The feed to the mill is through one hollow shaft, upon which the mill is supported and rotates, and the discharge is through the opposite axis. The power is applied through a countershaft equipped with a pinion, to a master gear surrounding the tube, either at the feed or discharge end.

The tube is equipped with a hard-iron or steel lining, which receives the wear and which can be renewed. This lining is either keyed into the mill or bolted in by means of bolts passing through the lining plates and shell. Both heads are also ordinarily lined with segmental lining plates of the same hard material. The grinding is accomplished by placing inside the tube such objects as hard flint, pebbles, cast- or forged-steel balls, cylinders or slugs, depending upon the ideas of the operator. Since the work done depends upon the energy of the grinding medium falling upon the rock to be crushed, it follows that the heavier substance accomplishes the most work and is the most efficient in use.

The tube mill originated in the cement industry, where an extremely fine product is required. Originally flint pebbles were employed as the grinding medium, and the mills were lined with silex or silex brick, a hard siliceous material. Recently, however, it has been found that the efficiency is greatly increased by using a hard-iron or steel liner and steel balls, the output per horsepower being far greater.

The feed is introduced through the hollow trunnion of the mill, in the case of dry grinding assisted by a helix cast as part of the trunnion lining, the screw carrying the

material positively to the interior of the tube. At the discharge end there may be a simple, plain opening through the trunnions, or a helix may be incorporated to facilitate discharge or the discharge may be by means of radial blades. This form of discharge is accomplished by placing a steel grating about 6 in. from the discharge end of the mill, the intervening space being occupied by a series of shelves or blades, extending radially from the periphery to near the trunnion opening. The blades deliver the ground material upon a cone whose base is against the grating, and apex extending into the trunnion opening, thus facilitating the discharge of the product. The advantages are that the discharge from a tube equipped with this system is essentially the same as a mill equipped with a peripheral discharge, but without its mechanical disadvantages. The finely ground material migrates rapidly toward the shell and discharge end of the tube, choking or plugging of the charge is avoided, and, in wet-grinding mills, it is possible to carry a low pulp level and low percentage of moisture. Crushing is more efficient, because each ball in falling, hits upon other balls covered with the material to be ground, and the blow is unimpeded. With the open-trunnion discharge, crushing is necessarily performed through a bed of pulp, which cushions the blow and dissipates the energy of the balls.

The proportions of ball mills, that is the diameter and length dimensions, have much to do with the results obtained from such mills. It is possible to so operate them that coarse, intermediate or fine product may be obtained. In this feature is one of the most favorable factors of the ball mill—it is very flexible and very adaptable.

The tonnage which may be put through a ball mill, and the fineness to which it will grind, depend on several factors, among which may be mentioned:

1. Hardness of the ore and size of feed.
2. Rate of travel through the mill.
3. Percentage of water.
4. Size of balls and number of balls.
5. Speed of mill.
6. Open- or closed-circuit grinding.
7. Type of mill, center or peripheral discharge.
8. Size of mill.

Some of these are variable or under the control of the operator; others are fixed, once the mill is running. Maximum mill efficiency is obtained when the greatest number of tons of ore is ground to the desired fineness in the shortest time, with the expenditure of a minimum amount of power and with the least wear on balls and liners. It is determined largely by the right combination of the factors above listed.

From the many excellent articles which have been written on current ball-mill practice, it is possible to gain information which will give a good idea of the correct combination necessary for the economic operation of proposed mills. No hard and fast rules can be formed, but, in general, the information gained will also prove useful in operating plants as giving a means of checking the work done by any particular mill, and of correcting existing faults.

1. *Hardness of Ore.*—This is beyond the control of the operator, but its size is not. Formerly, the tendency was to feed ore up to 4 in. in size. The more recent practice is to reduce the size of feed, for operators are determining the economic limitations of ball mills, and finding it preferable first to crush in secondary machines of approved type to 1 in. or finer.

2. *Rate of Travel.*—If the rate be slow, the ore is subjected to a greater number of blows, and is, of course, crushed finer. The tendency is, however, to feed fast and return the oversize, after classifying, for regrinding (see 6).

3. *Percentage of Moisture.*—This will vary with the character of the ore. Good practice is to keep the moisture low and the pulp thick, so as to coat the balls. This is conducive to less wear on balls and liners.

4. *Size of Balls.*—Balls from 1 to 7 in. in diameter are used. In a given mill the size of balls should be determined largely by the size and hardness of the ore. Best results seem to be obtained when the mill is loaded between three-tenths and two-fifths full.

5. *Speed of Mills.*—This is properly a direct function of the internal radius, and is something less than the "critical" speed. The "critical" speed of any mill is that which will make the layer of balls next to the shell cling to the latter as the mill revolves. Davis shows this to be $N = \frac{54.19}{\sqrt{r}}$ where N = revolutions per minute and r = radius in feet.

The modern practice is to operate at three-fifths to four-fifths of this speed. At four-fifths and above, the balls tend more to cascade, and the grinding is done largely by impact. At three-fifths the action is more of a rolling nature, and grinding is done by attrition. Operators are proving by decreasing the size of the feed that they may decrease the size of balls and increase the number in the mill, securing a greater grinding surface. As the power is directly proportional to the speed, the slower the mill turns, the less the power required. Also, the slower the speed the less the capacity, so a most efficient speed exists for any particular mill, and this must be determined by experiments on the ore under treatment.

6. *Open and Closed Circuits.*—The tendency has been to grind in closed circuit, crowding the ore through the mill, classifying, and returning the oversize. Good practice fixes this circulating load at from four to five times the feed. Closed-circuit operation tends to keep the mill up to full capacity, and is conducive to efficient working.

7. *Type of Mill.*—Opinion is divided as to the advisability of discharging the pulp through grates or overflowing through the trunnion without any diaphragm. The diaphragm makes the mill more sensitive to overloading.

8. *Size of Mill.*—The tendency is to the large-diameter mill; 9-ft. mills are the largest so far used.

The power requirements for ball tube mills operated at the proper speeds for best theoretical efficiencies and with various ball loads, has been calculated by Davis, and is shown in the accompanying table. These data are, of course, the result of mathematical computations and as such are correct, but in practice, some variations will be noted. The power requirements will vary in various operations. The figures shown in the table are high for general ball-mill work, but afford an intelligent guide to requirements. The proper speeds for best efficiency, also according to Davis, are shown in the appropriate table, p. 211.

In tube mill grinding, it is desirable to know the exact ratio between the tube-mill classifier circuit and the original feed. Tonnage samples with this end in view are not acceptable, as it is practically impossible to determine the moisture in a pulp from a sample large enough to represent the tonnage accurately. This ratio can be determined easily from the daily screen tests.[1]

Let,

x = Total original feed to classifier,
y = Tube-mill feed (classifier sands)
A = Per cent of finished size in the original feed,
(This will depend on the fineness of the overflow desired)
B = Per cent of finished size in tube-mill feed,
C = Per cent of finished size in tube-mill discharge, and
D = Per cent of finished size in classifier overflow.

[1] Geo. O. Deschler, *Eng. and Min. Jour.*, April 10, 1920.

Then,

Ax = Weight of finished size in the original feed,
By = Weight of finished size in tube-mill feed,
Cy = Weight of finished size in tube-mill discharge, and
Dx = Weight of finished size in the classifier overflow.

In regular operation, with the dry tonnage of the original feed equal to the dry tonnage of the classifier overflow, at any one instant in the cycle the weight of the finished size in the material going to the classifier will equal the weight of finished size in the classifier products. Therefore,

$$Ax + Cy = Dx + By$$

$$y = \frac{(D - A)}{(C - B)} x$$

What is required, then, for determining this ratio, is the percentage of undersize from the limiting screen, on the four products as stated above. A few examples from practice will serve as illustrations. In the following work 15 per cent + 65 mesh, Tyler standard screen, was desired in the classifier overflow.

Example 1.—Oversize in the overflow approximately as desired.

	Per Cent
−65 mesh in the original feed	38.0
−65 mesh in the tube-mill feed	27.8
−65 mesh in the tube-mill discharge	49.2
−65 mesh in the classifier overflow	87.0

Applying the equation, $y = 2.29x$.

Example 2.—Oversize in the overflow greater than desired.

	Per Cent
−65 mesh in the original feed	29.8
−65 mesh in the tube-mill feed	30.4
−65 mesh in the tube mill discharge	45.8
−65 mesh in the classifier overflow	80.4

By substituting, as before, $y = 3.28x$.

With a higher allowable amount of oversize in the overflow, the ratio of tube-mill feed to original feed increases, under the stated conditions.

Example 3.—Oversize in the overflow less than desired.

	Per Cent
−65 mesh in the original feed	29.6
−65 mesh in the tube-mill feed	35.4
−65 mesh in the tube-mill discharge	81.8
−65 mesh in the classifier overflow	94.6

Using the equation, $y = 1.40x$.

As the amount of oversize in the overflow is decreased, the ratio of tube-mill feed to the original feed decreases, under the stated conditions.

It is the general practice in ball-mill grinding to run the feed direct to the mill instead of to the classifier, as in the cases considered above. With this arrangement, let

x = Weight of original feed to mill,
y = Weight of classifier sands,
A = Per cent finished size in original feed,
B = Per cent finished size in classifier sands,
C = Per cent finished size in mill discharge, and
D = Per cent finished size in classifier overflow.

Then

Ax = Weight of finished size in original feed,
By = Weight of finished size in classifier sands,
$C(x - y)$ = Weight of finished size in mill discharge, and
Dx = Weight of finished size in classifier overflow.

As before, the weight of finished size in the material going to the classifier will equal the weight of finished size in the classifier products, so

$$C(x + y) = By + Dx$$

or

$$y = \frac{(D - C)}{(C - B)} x$$

The ratio of circulating load (R) to original feed would be

$$R = \frac{x + y}{x} = 1 + \frac{Y}{x} = 1 + \frac{(D - C)}{(C - B)}$$

Example 4.—As an illustration, suppose the figures are the same as in Example (1) except the percentage of undersize in the tube-mill discharge.

	Per Cent
−65 mesh in the original feed	38.0
−65 mesh in the classifier sands	27.8
−65 mesh in the mill discharge	59.4
−65 mesh in the classifier overflow	87.0

Substituting in the equation, $R = 1.87$.

Properly speaking, the circulating load is that part of the load which circulates, or returns to the starting point. In the case of the feed going direct to the ball mill, the circulating load is made up of the classifier sands only, as that is the only part of the original feed which completes the cycle. Following this concept, the ratio (R) of circulating load to original feed would be $R = \frac{Y}{x}$ instead of $1 - \frac{Y}{x}$.

Horsepower Required per Foot of Mill Length

Internal diameter of mill, feet	Proportion of mill volume occupied by charge					
	0.1	0.2	0.3	0.4	0.5	0.6
1	0.0023	0.0099	0.022	0.042	0.065	0.093
2	0.0255	0.1100	0.254	0.460	0.730	1.040
3	0.1087	0.4630	1.070	1.940	3.070	4.380
4	0.288	1.2400	2.870	5.220	8.250	11.780
5	0.640	2.7200	6.290	11.430	18.070	25.740
6	1.200	5.1600	11.880	21.470	34.140	48.690
7	2.060	8.8500	20.430	37.060	58.610	83.530
8	3.260	14.1200	32.540	59.080	93.460	133.320
9	4.960	21.3200	49.090	89.310	141.240	201.330
10	7.130	30.8100	71.060	129.000	204.070	291.100

In this table the mill is assumed to be operating at the most efficient speed, as shown in the following table, and the charge is assumed to weigh 325 lb. per cubic foot.

SPEED OF MILL FOR BEST THEORETICAL EFFICIENCY*

Internal diameter of mill, feet	Proportion of mill volume occupied by charge at r					
	0.1	0.2	0.3	0.4	0.5	0.6
1	59.15	60.45	61.82	63.36	65.10	67.01
2	41.88	42.75	43.72	44.81	46.04	47.40
3	34.19	34.90	35.69	36.58	37.58	38.69
4	29.61	30.22	30.91	31.67	32.55	33.51
5	26.48	27.04	27.64	28.34	29.11	29.96
6	24.18	24.69	25.24	25.88	26.58	27.36
7	22.38	22.85	23.36	23.94	24.60	25.32
8	20.94	21.38	21.86	22.40	23.02	23.70
9	19.74	20.15	20.60	21.13	21.70	22.34
10	18.72	19.12	19.56	20.04	20.59	21.20

* Davis, Trans. A. I. M. M. E.

Probably the most important difference of opinion in ball- and tube-mill operation is in the proper size of feed to be supplied. Extremes are represented by the Inspiration plant, Arizona, on one hand, where 3-in. material is reduced at one operation to −48 mesh in a large diameter mill, and the Canada Copper Corporation, on the other hand, where crushers and rolls are used to bring the ore down to 10 mesh, after which two-stage ball-mill reduction carries it to −100 mesh. The tendency of modern operations is to save power by crushing fine before introducing the material into the ball mills.

An important modification of the tube-mill is the Hardinge mill, in which the discharge end is cone-shaped. The Hardinge modification is based on the theory that this construction leads to a classification of balls in the mill, the small balls or pebbles taking a position near the discharge where the material being treated is also finest. Theoretically, then, the material being pulverized and the pulverizing medium are proportioned in size in this mill with a resulting operating economy.

CRUSHING PRACTICE

Primary Breaking.—Primary breaking, in the usual acceptance of the expression, means the first stage of rock reduction, in which the rock from its original source is given its first operation of size reduction. Primary breaking usually refers to the first operation at the particular installation where the work is being done, consequently it may vary in its requirements from the ability to receive pieces 6 or 7 ft. in diameter to machines required to accept only 8- to 12-in. pieces. In general, however, it is most convenient to consider primary or coarse breaking as the field in which the rock or ore directly coming from quarry or mine is fed into the machinery providing the first breaking state.

In large quarrying operations, the rock as it is blasted from the solid, ordinarily contains pieces too large to enter any breaker yet built. These pieces have to be drilled and blasted so as to secure sections that will go into the machines available. Jaw breakers have been built capable of receiving 7-ft. pieces of rock, and gyratory breakers capable of receiving 5-ft. pieces. There is no reason why larger machines should not be constructed, except that they are very expensive and their tonnage

capacity is so great that it can very rarely be utilized. However, it is usually considered advisable to use a primary-breaking machine large enough to receive the largest piece ordinarily encountered in order to avoid the extra expense incurred in drilling and blasting. The labor cost of such operation is often so great that it would more than overbalance the extra installation cost of a larger primary breaking machine.

The large machines used for primary breaking reduce to rock to pieces 5- to 10-in. cube, a size at which they are properly sized for delivery to secondary machines.

Both jaw crushers and gyratories are used as primary breakers, various operators having different opinions about the relative merits of the two types for such service. Many of each are in use in various parts of this country. Their relative advantages have already been discussed under the head of gyratory and jaw crushers in the first section of this paper.

Intermediate Crushing.—This stage of rock reduction is the one in which rock is taken from the primary crushers and again reduced in size. In a few cases, such as in plants where limestone is crushed for blast-furnace flux, there may be only one stage—the primary, in which the stone is produced at the proper size in one operation.

In all crushing, no matter how the crushing machine is set, there is always a large percentage of the crusher product that is much smaller than the minimum crusher opening. The crusher discharge will vary in size all the way from fine dust to pieces that will just pass the crusher opening. The desired size is separated from the aggregate by screening, usually in a rotary screen for coarse stone and in shaking or vibrating screens for the finer sizes. When there is but one crushing stage, the crusher product is screened and classified so as to produce products that are marketable, thus avoiding waste.

When secondary, or intermediate crushing is to be performed, it is usual practice, and proper, to recrush only that portion requiring further reduction. This is accomplished by passing the product of the primary machine over a screen, sending only the oversize to the secondary crusher, allowing the material already fine enough to bypass the secondary machine and proceed to the next stage, or to the one where the product can be utilized in its existing form, or to a machine appropriately designed for its further size reduction.

Secondary crushing, as has already been pointed out, may be performed in machines of various types, but for such work, the gyratory principle is being generally preferred.

Granulating.—In many industries, it is necessary to produce material in granular form—a form in which the particles are considerably smaller than is ordinarily produced by secondary crushing, and not so small as when a pulverulent product is required. The requirements may be such that the product wanted is anything from 40 mesh to 10 mesh, the mesh meaning the number of apertures per lineal inch. Reduction of this character can be performed in any one of many different ways, but according to usual practice, is accomplished by swing-hammer machines, rolls or ball tube mills.

When the swing-hammer mill is used for this purpose, a fine grid screen is used, and the product of the machine is screened to secure the classification desired. Mills of this type do not produce a uniform material, in-so-far as size is concerned, and for that reason they are not popular for use where uniform particles are desired with a minimum of both oversize and undersize.

Crushing rolls are very widely used for such work and are usually very satisfactory. The distance between the roll faces can be definitely set to the size of particle desired and there can be no oversize. There will be, however, a large percentage of undersize because roll crushing is the result of direct compression, and consequently a good deal of fine dust is produced. Another objection to the use of rolls is that the crushing

faces do not wear evenly, but soon corrugate, making it impossible to maintain a fixed discharge.

Ball-tube mills may be used very efficiently for granulating. The reduction is accomplished by impact, if the device is properly operated, so that a minimum of undersize is produced. By using a short mill and a heavily forced feed, classifiying the mill product and returning the oversize for further grinding, a uniform output is secured, together with minimum power expenditure. In such work it is necessary properly to proportion the size of the ball used to the size of the piece to be broken.

Pulverizing.—In pulverizing, the work to be done is usually the reduction of particle size so that the product will pass through a wire screen having 100 apertures per linear inch or even smaller. Such service can be performed in ring-roll mills, machines of the Huntington type, chilean mills, grinding pans, grinding disc, etc., but the tube mill, charged either with steel balls or flint pebbles, has by far the greatest use and is considered the most effective and efficient. In cement making, where a very fine product is obtained by dry grinding, tubes up to 8 ft. in diameter and 26 ft. long are used, making a product that is nearly all under 200 mesh and varying from that down to material that would pass a 1000-mesh screen, if such a screen could be made. Pulverized coal for fuel is so produced, and in mining the production of finely divided ores for cyaniding, flotation, etc. is so produced.

A modification of the tube mill loaded with balls or pebbles is one that is loaded with steel rods (Marathon, Marcy, Cole and others). This arrangement acts as a multiple roll, grinding the material between the slowly turning surfaces. The machine is said to give excellent results, particularly in granulating. The charge of rods is carried above the horizontal axis of the mill, thus tending to balance it and reduce the amount of power consumed. Machines of this type have been in use experimentally for several years and recently have been installed on a large scale in one or two mining installations.

Pulverization is the most expensive part of all rock reduction, more energy must be expended to accomplish it, and consequently this stage has received, and should receive, a maximum of study and attention.

MISCELLANEOUS CRUSHING AND GRINDING MACHINES

The stamp, either falling under the influence of gravity only, or driven by steam, has been of great importance in the development of gold and copper mining, but has no use, so far as known in other industries, except metallic junk treatment. It is an uneconomical method of crushing ore, and any one interested can find a detailed description and discussion in Richards "Ore Dressing."

The first crushing machine, after the war club, was the mortar and pestle. Except as a laboratory implement, this has become more of a mixing device and is described under "Mixing and Kneading." **The bucking board** of the assayer may be considered a mortar that has been flattened out.

The so-called drug mill consists of a cone revolving within another cone (or both surfaces may be curved) without eccentric motion, wherein it differs from the gyratory crusher. These surfaces usually have corrugations on them so that comminution is mainly a matter of shredding and tearing rather than of crushing or abrasion.

The instrument is of rather wider utility than is claimed by the manufacturers, even such material as copper drillings being shredded up by it, though in such cases the cups and cones furnished by the makers might profitably be replaced by those of chrome or manganese steel. For certain materials I have also found better results were obtained by running the cone in the opposite direction from that indicated by the manufacturer.

Symons Disc Crusher.—The range of this machine, shown in Fig. 10, is in the larger sizes, crushing from 6 in. to 1½ in. It is claimed that the largest sizes will crush to an inch but all factors considered the higher figure would be the practical minimum size which it will deliver. The two discs *A* and *B*, are held in place by

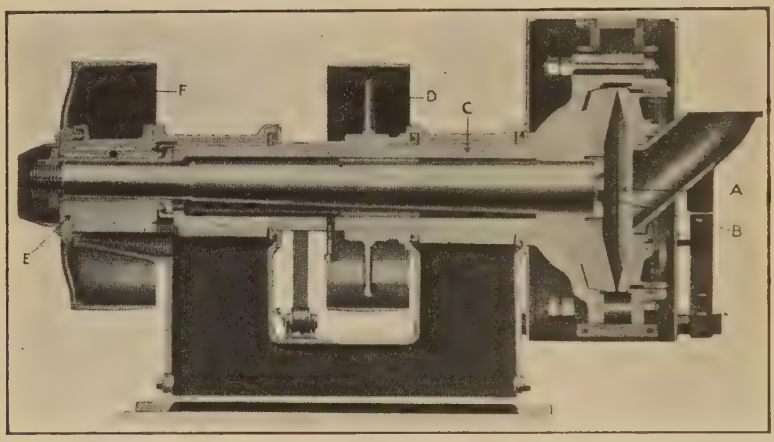


FIG. 10.—Symons disc crusher.

two shafts one of which is hollow and contains the other. The large ball and socket termination of the inner shaft allows the inner shaft actuated by the eccentric *E* to partake of a rocking motion so there is constantly changing set between the two discs and they are constantly changing the angle they make with one another. The inner shaft gyrates in a direction reverse to the rotation of the disc *A*. The material entering the feed spout is thrown out to the periphery of the two discs by centrifugal force. If a fragment is caught near an outlet point of the discs which happens momentarily to be a close together it will be released and thrown out by centrifugal force after the width of exit is increased under rotation. If a larger fragment is caught between the discs when they are at or nearly at their point of greatest set it will be carried around and crushed when the discs come closer together and will then be thrown out into the enclosing hopper. A magnetic tramp-iron separator is necessary ahead of a disc crusher. It is virtually a gyratory crusher turned on its side.

Disc grinders are also made in which the discs are parallel and pulverizing takes place through attrition between the faces, which may be either plane or grooved. After all, they are a variation of the old fashioned buhr-stone, set vertical instead of horizontal, and of steel or cast-iron instead of stone.

For the shredding of fibrous materials, paper, rags, etc., the beater of the paper makers may be employed. As may be seen from the illustration, this consists of a heavy wheel with bars set in (practically a roll with corrugations

parallel to the axis) revolving over a bed plate which is also corrugated. The material is thrown up by the beater wheel over a barrier and then comes around a race course and under the beater again.

Sizes and Capacities of Large Disc Crushers

	Inches	Inches	Inches
Size of crusher	48	36	24
Opening in elliptical feed spout	11¾ by 17	9¼ by 14½	7 by 10⅝
Width between discs at feed spout	8	5	3¼
Capacity, tons per hour to 1½ in., approximately	40	25	15
Horsepower required	65	40	25

Aids to comminution are of various sorts. Gummy or resinous materials can often be ground when chilled when they resist all attempts at subdivision at ordinary temperatures. Occasionally a material which is to be added at a later time greatly increases the readiness with which some other material can be ground. Practically all the waterproofing agents used with cements can be readily ground after adding some of the cement, where they cannot without it. The excipients of the druggists come in this category.

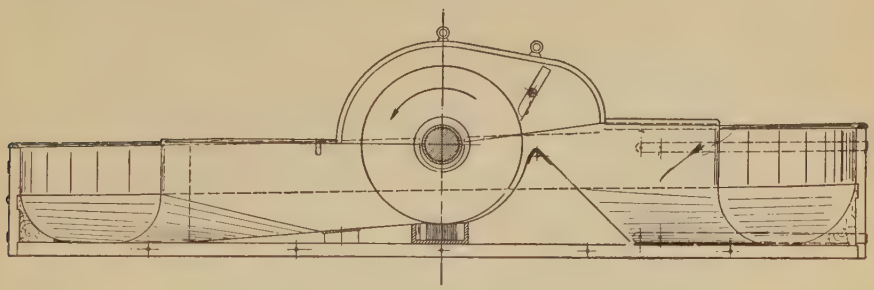


Fig. 11.—Longitudinal section of beater.

Work Done in Rock Crushing.—It is important, in order to compare the work accomplished by various machines, to have some basis upon which such comparisons may be founded; There are two basic laws that have had attention from students of the subject—those of Rittinger and Kick. Rittinger's law states that the power required for reduction is proportional to the increase of surface. Kick's law states, in effect, that the power required varies as the volume or weight.

Both of these suggestions cannot be correct, since the Rittinger theory calls for the expenditure of approximately 27 times as much power to reduce 1-in. cubes to 200 mesh as is required by the Kick theory. A series of tests and experiments made by Prof. John W. Bell, of McGill University, led to the conclusion that the Rittinger theory was more nearly correct and could be used as a basis for comparisons. A paper by Arthur O. Gates, in the *Engineering & Mining Journal*, May 24, 1913, gave a practical method of illustrating these comparisons and is reproduced herewith.

Crushing Surface Diagram.—In view of the importance of crushing efficiencies in various industries, and the frequent discussion taking place on this subject, the diagram which may be called the crushing-surface diagram is submitted as a means of comparison and study of crushing conditions. The scheme was discovered during the summer of 1910, when trying to understand why one tube mill did not do the work expected of it on the basis of what another was doing on "a similar ore just as hard."

First it should be stated that this diagram is based on Rittinger's law, which is interpreted to mean that the work done in crushing is proportional to the surface exposed by the operation, or better expressed for this purpose, the work done on a given mass of rock is proportional to the reciprocal of the diameter of the final product, assuming that all the mass has been reduced to one exact size, which is only theoretically possible.

Kick's law is frequently referred to in connection with this subject, especially since the publication of H. Stadler's[1] work, and is thus expressed: "The energy required for producing analogous changes of configuration of geometrically similar bodies of equal technological state varies as the volumes or weights of these bodies." This law does not apply so much to crushing as to deformation of bodies before rupture takes place.

In Fig. 12 are represented two particles of ore "of equal technological state" shown as cubes between the faces of a crushing or testing machine. Assuming the theoretical mesh, equivalent to the reciprocal of the diameter, and using concrete values, we have a 100-mesh particle with eight times the volume of the 200-mesh particle, and with an area per face four times that of the 100-mesh particle. The dimensions are as 2 to 1, and the bodies being similarly deformed within the elastic

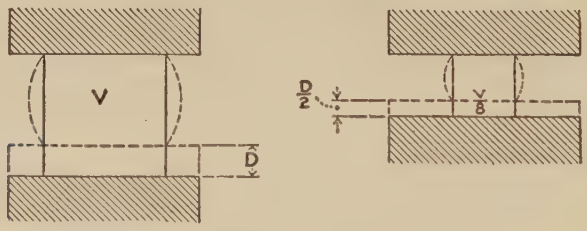


FIG. 12.—Kick's law.

100-mesh cube	200-mesh cube
Area one face = A	Area one face = $\frac{A}{4}$
Energy = FAD	Energy = $F\frac{A}{4}\frac{D}{2} = \frac{FAD}{8}$

$\therefore$ Energy proportional to volume.

limit without fracture, the energy that must be applied in each case to produce this deformation is the product of the average resisting force per square inch, the same in both cases, by the area worked against and by the distance through which this average force works. As shown in the figure, in this particular instance the energy absorbed is proportional to volume and it can be similarly shown for the general case. On the gradual release of the external pressure the energy absorbed is given back to the machine producing the deformation and the body returns to its original shape. It should be noted that the body has been deformed only by a gradually increasing pressure, the first increment of deformation not requiring so much pressure as the last. In case the body has been deformed beyond its elastic limit either the whole mass of particles has been reduced to the molecular state by the freeing of their bonds with

[1] *Trans. I. M. M.*, Vol. 19, "Grading Analyses and Their Application."

adjacent particles, which never happens, or fracture takes place along a few surfaces by the breaking down of some of the weaker bonds, and the new particles thus formed are free to resume their original shape in-so-far as they are not held between the machine surfaces. The energy given up by them is probably used in some sort of lever action in making fracture planes. So the energy absorbed according to Kick's law does not stay in the particle after pressure is released and; therefore, this law does not govern to any great extent the amount of energy absorbed in crushing.

Reference to Fig. 13 will explain how Rittinger's law applies. Supposing that it were possible to hold similar cubes between the two offset faces as shown and that

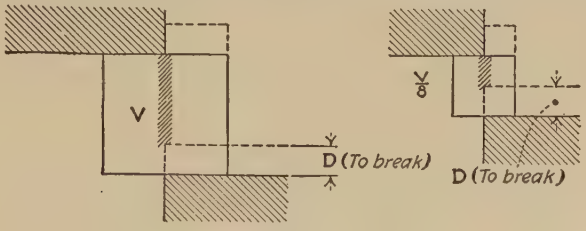


FIG. 13.—Rittinger's law.

100-mesh cube	200-mesh cube
Area one section = A	Area one section = $\frac{A}{4}$
Average resistance to shear per square inch = F	
Energy = FAD	Energy = $F\frac{A}{4}D$
Surface produced = 2A	Surface produced = $2\frac{A}{4}$

∴ Energy pro portional to surface.

forces were applied until the deformation shown by the dotted lines was obtained, it will be seen that only the molecules along the vertical center line are stressed and deformed, the mass of the cube away from this surface receiving practically no pressure or deformation. The energy in this case required to produce rupture will be the product of the average resistance to shearing per square inch by the area along which rupture takes place, and by the distance the two offset faces move together. The average resistance to shearing is a variable quantity, as the deformation increases up

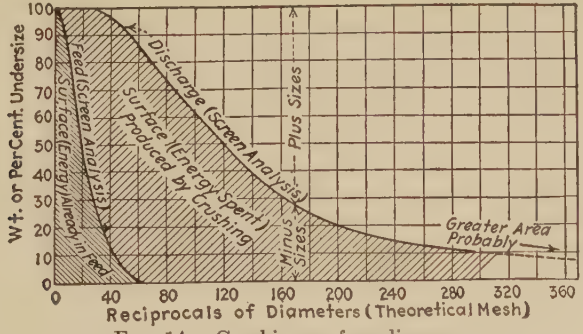


FIG. 14.—Crushing surface diagram.

to rupture. To reduce to cubes, this amount of energy must be multiplied by three, as three similar fracture planes must be made to produce cubes. And it will not be hard to see that the distance through which the offset faces must move in either case must be the same and not proportional to the thickness of the piece. To break the molecular bond between adjacent particles would require the same movement regardless of the thickness of the piece.

It will be noted that if eight of the half-diameter cubes be sheared to produce cubes of half their size, the new surface presented will be double that formed when the single large cube is sheared into half-size cubes, and also that the energy required in the case of the smaller cubes is double that required in the case of the larger cubes. This should demonstrate that energy applied to crushing is proportional to the surface produced.

In this scheme we start from an infinite mass, say the side of a mountain, and consider that particles are sheared off in the form of cubes, each particle having new surface added to it equal to that exposed before the new shearing operation was started. This eliminates the "minus one" part of the equation given in Richards' "Ore Dressing" on the subject. For example, if we cut off from a cubical corner of this infinite mass 100 slabs 1 in. thick and 100 in. square, and then cut the slabs up into 1-in. cubes, 6,000,000 sq. in. of new surface is produced. If instead, 200 ½-in. slabs

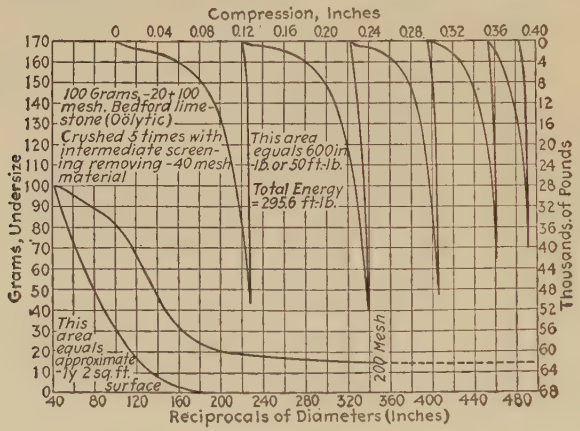

Compression, Inches
0 0.04 0.08 0.12 0.16 0.20 0.24 0.28 0.32 0.36 0.40
Grams, Undersize
Thousands of Pounds
100 Grams, -20+100 mesh, Bedford limestone (Oölytic) Crushed 5 times with intermediate screening removing -40 mesh material
This area equals 600 in-lb. or 50 ft-lb.
Total Energy = 295.6 ft-lb.
This area equals approximately 2 sq. ft. surface
200 Mesh
Reciprocals of Diameters (Inches)

Fig. 15.—Diagram of crushing tests.

were cut from the same corner and likewise cut up into ½-in. cubes, the area would be 12,000,000 sq. in., just double the surface of the 1-in. cubes. This shows that the surface produced on equal masses of rock is proportional to the reciprocal of the diameters, as can be proved in a similar manner for any sizes. Also the surfaces of two lots of the same sized particles will be proportional to their values, and if of the same substances, proportional to the weights of the lots.

From the above we see that the energy absorbed by a lot of given sized particles is proportional to the product of their surface by their weight, which can be shown graphically by an area. And if we have a series of these areas, placed side by side, and representing the summation of the energies of the different sizes produced by a crushing operation, the total area is proportional to the energy expended on the rock. And when the cumulative weights of the different sizes are plotted consecutively, as in Fig. 14, the area between the sizing-analysis curve and the zero lines is proportional to the work expended on the rock in breaking it down from the infinite mass. And then if further crushing takes place on all or part of this rock and the new-sizing analysis is plotted, the area between the two curves is proportional to the further work done. This is the crushing-surface diagram.

For rough work, comparing the work of one day with that of another, the screen analysis may be plotted, mesh against the percentages, and areas compared, etc. For accurate work, such as the determination of constants for different rocks, or for the comparison of the efficiencies of different crushing machines, etc., the actual

reciprocals of the diameters should be plotted. In fact, it is no hard matter to locate on the diagram the reciprocals of the diameters for the screens used; or better still, screen manufacturers could be sufficiently induced to furnish pads of coördinate paper, on which are drawn these lines corresponding with their standard sets of screens.

Two uses are already suggested; the first, determination of crushing resistances of various rocks, and second, the determination of crushing-machine efficiency. Only with a knowledge of the former, can comparisons be made of the operation of two machines at different places. And I believe that this diagram will make necessary some revision of the constants for the crushing resistance of rocks. All tests of rocks have been made on single pieces and the crushing strength determined from an average of a number of tests. Where surface has been determined it is only that of the larger pieces involved in the fracture. No calculations based on the surface of the very fine particles have been made. One thing which will be noticed in nearly every

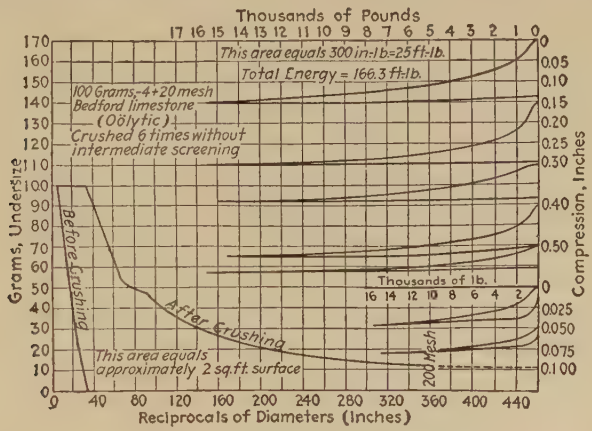

Thousands of Pounds
17 16 15 14 13 12 11 10 9 8 7 6 5 4 3 2 1 0
This area equals 300 in-lb=25 ft-lb.
Total Energy = 166.3 ft-lb.
100 Grams, -4+20 mesh Bedford limestone (Oölytic) Crushed 6 times without intermediate screening
Grams, Undersize
Compression, Inches
Before Crushing
After Crushing
Thousands of lb.
16 14 12 10 8 6 4 2
This area equals approximately 2 sq.ft. surface
200 Mesh
Reciprocals of Diameters (Inches)
0 40 80 120 160 200 240 280 320 360 400 440

Fig. 16.—Diagram of crushing tests.

screen analysis that is plotted as a crushing-surface diagram, is that the curve is of the form of a hyperbola, indicating a strong probability of the presence of particles of 1,000 and even 10,000 mesh. Some allowance for the probable area of these very fine particles must be made, and when that is done it will very likely be found that there is a greater area in the −200-mesh particles than in the +200-mesh material. There is no question but that the surface of the −200-mesh particles must be considered in crushing calculations. The neglect of this surface, which usually is not wanted, makes any reasonable comparison impossible between crushing operations that are not very much alike. The agriculturists in the study of soils do not hesitate to measure and discuss particles of clay as small as 0.0002 in. in diameter, and so I see no reason why milling operators should avoid these sizes and talk of colloids as soon as they get past the 200-mesh line. The use of logarithmic paper can be invoked to find out about the −200-mesh particles as I may show later.

Other uses for this diagram should be in the cyanide plant, where time of solution depends upon surface exposed, and the amount of surface is a factor in filtering operations. In the cement industry, surface must play an important part, for each particle of stone or sand or fine material must be coated with cement particles before a good bond can be made. The amount of cement could be made proportional to the area of the diagram for the material used.

In concentrating-mill operations, surface is fully as important in the concentrating department as in crushing operations. It is surface that limits the tonnage that can

be put through a given machine, only a certain number of layers of particles can be treated at one time on a given surface, which explains the small capacity of fine concentrating machines. Based on surface, the distribution of feed to different machines can be made more efficiently. Also the character of the crushing operations can be modified to give the largest reduction possible with the minimum production of useless fine surface.

The two diagrams, Figs. 15 and 16, are made from the logsheets of the tests, all records of crushing pressures being plotted directly on the sheet against the movement of the faces of the machine. The triangular curved areas are the record of the work on the machine, and are a measurement of energy. Each crushing movement was continued until the pressure was rapidly going up with little movement of the faces, then the pressure was gradually reduced and we got the return curve showing that some energy was being given up by the rock. The machine was then opened, the finer sizes screened out and the coarse returned and crushed again. After the crushing was complete as far as was considered advisable, the screen analysis was made and plotted as the weights were determined. One hundred grams was used it each case, its original screen analysis being the first thing plotted on the paper.

Without going into the calculations in detail, the energy absorbed in the case of Fig. 15 was 166.3 ft.-lb., while in the case of Fig. 16, the energy absorbed was 295.6 ft.-lb., about 80 per cent more. And yet the area of the surface under 200 mesh was greater in the case of the former. The only way to account for the energy is to go beyond the 200-mesh line, and doing this, using logarithmic paper and assuming that the curve continues its law indefinitely among the smaller particles, I find that out to 5,000 mesh for Fig. 15 the energy is equal, per unit area, to that in Fig. 16 out to 11,000 mesh. Both are calculated to have one-half of 1 per cent at those sizes.

Screening and sizing of the products from the various grinding machines will be found treated in the following section.

SECTION VI

GRADING AND SCREENING

By Edward S. Wiard[1]

Definitions.—This and the succeeding two sections deal with the differentiating of mixtures of materials, solids, liquids and gases by mechanical means or machinery to the end of obtaining one or more portions of the mixed masses in a form useful in the arts. For the purposes of general definition mixtures of solids, gases and liquids will be considered a mass in entirety, even though the separation for example of a solid from a gas could scarcely be denominated a differentiation. The word differentiation is used here in the sense of splitting up a more or less homogeneous mass into components having but slight physical differences from the parent mass. Such differences while slight from the physical sense are often very great in the commercial sense. The separation of the "outside" and the "thickness" from the "belly" or portion next to the tree of the bark of *Quercus Ilex*, a problem the real solution of which still awaits the inventor, makes wood, waste and worthless, and the article of commerce known as powdered or granular cork a commodity of great value.

It is necessary at the outset to distinguish between devices which split or separate a mass into two parts one part being usable and the other being either worthless as in the example cited above or as is most usually the case capable of being further subjected to processing to get an additional yield of usable part, and devices yielding more than two products out of the original mass. In the preparation of the natural amorphous graphite from the Santa Maria mine, Sonora, the raw material after grinding passes to separators of the Raymond type. These yield impalpable graphite and coarse graphite which is returned to the grinding machines. The process is continuous. In the abrasives industry various sized sands are prepared for gluing to paper and cloth by batteries of screens and furnish an example of a device yielding a plurality of products.

Grading.—The definition of this word is "sorting out or arranging in order according to size, quality, rank, degree of advancement, etc." While open to objection this term is the most comprehensive to be found for the operations of assorting or sorting, blowing, classifying, screening and sifting. That these terms are used indiscriminately by inventors a reference to the *Patent Office Gazette* will disclose.

For the subject matter of this chapter there will be scant necessity for mentioning the terms assorting and sorting. These terms should be used for indicating the operation of removing from a mass of units usually by hand some which are strikingly different from the balance as in the operation of hand sorting ore from waste. The term assorting should be used to indicate the grading of material from coarse to fine by the action of streams and bodies of water. The term sizing has been omitted not because it is not good usage as a synonym for screening but to avoid confusion with the operation of giving surfaces a coat of size or gelatinous wash.

[1] Consulting engineer, 409 Boston Building, Denver, Colo.

The term classifying is used for the grading effect obtained on subjecting a granular or powdered mass to the action of a current of liquid in a tank or other device for giving the same effect. This produces a gradation from coarse to fine in the enclosing tank or other kind of apparatus used. Any length of the settled particles can be taken as a grading and the contamination of the coarse particles by the very fine ones due to eddying currents or the clinging of the fine particles to the coarse ones in the descent can be obviated to some extent by constricting the discharge openings placed near the bottom of the settling apparatus used and introducing through such constricted opening an upward rising current of liquid. The term can also be used for settlement and siphoning operations without flow and where the time of settlement is progressively increased. This is the standard method used in the abrasives industry for producing very fine powders.

Separation.—This is a term much used by inventors synonymously with the others given. In this book its use is confined to operations and devices for separating solids from solids, solids from liquids, solids from gases and liquids from liquids. The great commercial importance of the centrifugal separator sanctions the use of the heading to the operations performed by this machine or operations where two products issue or arise, one often being waste. In addition to centrifugal separators, filtering apparatus will be considered under this head, also the so-called "classifiers" which have been developed in Western ore treatment practice. These devices separate in water ore which has been finely ground into sands and a finely divided part known in ore treatment as slime. The name classifier applied to them is firmly entrenched. Dry separators for splitting dry materials into fine and coarse parts will also be described under this head. Discussion of separation will be found in Section VII.

Concentration.—Under this head is comprised the operations of differentiating a mass of ore, earthy material or other substance according to the specific gravity of the components of such a mass. Usually one or more merchantable products of greater specific gravity than the original average are obtained from the parent mass and the balance of it of low specific gravity is the refuse, waste or tailing. In concentrating operations on some substances such as coal from the mine the merchantable product has less specific gravity than the refuse. A short definition of concentration would be "The art of enriching ores earthy or other substances by machinery." The full discussion of concentration is given in Section VIII.

Grading.—The processes under this head have mostly to do with fragmental granular or powdered masses of solid material and the means of splitting up a comminuted mass according to size. Means for reducing them to this state from larger unit masses will be found discussed under the head of "Crushing and Grinding."

Some reasons for dividing or splitting up a loose or broken mass of material into a number of sizes will be gathered from the following examples of the need of this operation in certain industries. In the use of abrasives the aim is to produce a smooth or polished surface from a roughened one. To gain this the roughened surface must be reduced progressively by abrasive grains whose diameter is roughly proportional to the depth of the projections of the roughened surface. Quite evidently while the abrasive grains are tearing away the projections of the surface, they are grooving the smoother portions to an amount proportional to the size of grain used, though to a lesser degree. On this account, if the ultimate aim is a smooth or polished surface, the abrasive must be used in graded masses starting with a coarse size and ending with a fine.

For human and animal consumption salt is graded to obtain varying rates of solubility. For household use a quickly dissolving product is desired, while for layering meat the salt is desired in a condition such that it will dissolve slowly. This requires that the salt for household purposes be fine but not so fine that it loses its crystalline character for it would cake too readily in the receptacles for holding it in a damp climate. If fine salt were used for preserving meat it would quickly dissolve in the moisture of the flesh and run to waste permitting the entry of decaying agencies. A very coarse salt is consequently used for layering meats, intermediate sizes between this and the finest being used where greater rates of solubility are desired.

Miscellaneous Modes of Grading.—A means for separating shrunken and discolored beans from perfect white beans is disclosed in a recent patent (U. S. Patent 1,080,088, Dec. 9, 1913). By this patent means are provided for bringing

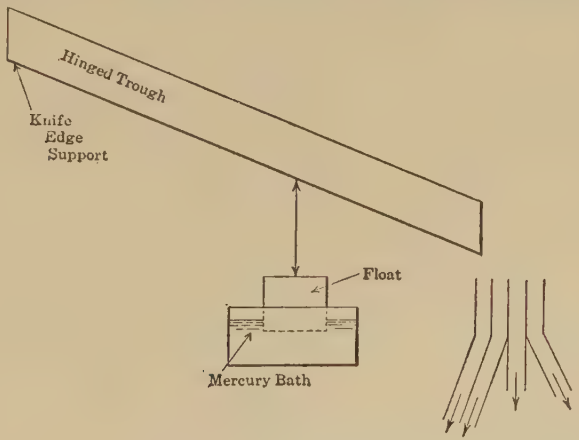


FIG. 1.—Grading by weighing by means of a hinged spout.

the beans singly to a position in front of a hinged spout. Means are provided for reflecting all the light from the beans onto a selenium cell which consists of narrow strips of selenium joining broad brass plates the whole forming a portion of an electric circuit. If the hinged spout be supported by the attraction of a magnet which is part of the selenium cell circuit, then when the perfect beans pass through the apparatus, since the resistance of the circuit will be diminished from the greater light impinging on the cell, the spout will be raised to the highest point and if a receptacle is placed in the proper position below the edge of the spout the perfect beans will fall into it. If other receptacles or compartments are placed behind the first a gradation can be obtained from perfect white beans to dark, shrivelled beans. Greenish-yellow rays lower the resistance of selenium the most. The device described may offer a means of grading according to color and may have some application in concentrating ores.

Grading by One Dimension Only.—A silk-sorting machine is one for grading threads according to thickness and winding them upon the proper bobbins. The proper bobbin is presented to the thread by the action of a lever which is governed by the thickness of the thread passing between gage rollers.

Grading by Weighing.—Grading by weighing is only adapted to relatively large individual pieces. Each part of such a device must have means for feeding the individual piece singly and the interval of time between pieces must be suffi-

cient to completely weigh and dispose of a piece before another passes through the mechanism. The transportation element must not influence or interfere with the weighing element. A simple mode of obtaining grading by weighing would be to have a spout hinged at one end and supported by scale beam connections at the other. Either of the devices shown in Fig. 1 or Figs. 2 and 2A would furnish the proper weight opposing thrust, that of Fig. 1 being a mercury float and that of Figs. 2A and 2 being a sine curve suspended from which at point *A* is a strap which is connected below with the free end of the scale beam connections. Figure 2A shows the adjustable weight opposition means. This is fastened by means of the $\frac{5}{32}$-in. clinching bolts to the same shaft that carries the sine-curve block. As the scale beam connections are depressed the counterweight swings out to a balancing point. For equal increases in increment of weight there are equal angular movements of the shaft and weight and by hinging the spout to the shaft and with suitable connections with the scale beams the spout will depress through equal angles for equal increments of weight of pieces passing down it.

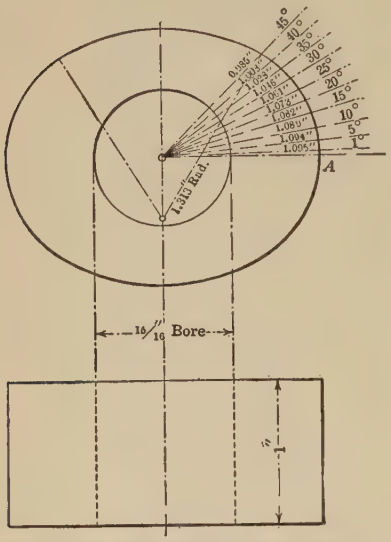


FIG. 2.—Sine curve block for grading by weighing.

Grading by Blowing.—In grading by blowing the air most commonly is introduced at the feeding point and from this point the air is expanded into a chamber of increasing cross-section. The air may be exhausted at the end of

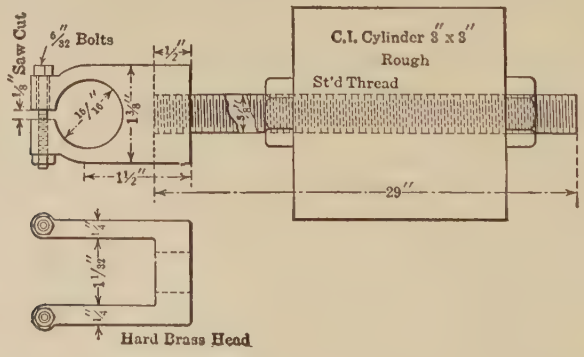


FIG. 2A.—Adjustable weight for sine-curve block.

such a chamber, particularly if the very fine dust reaching the end of the grading chamber is to be forced through a dust-collecting apparatus. If the very finest dust is to be collected in dust chambers which must be of large volume as compared with the grading chamber then the aim will be to have the velocity of the air current at the end of the grading chamber as low as possible.

At regular intervals in the expansion chamber gates must be placed to receive the gradings which will be more or less of a size inversely proportional to the mean cross-sectional area of the grading chamber within the draw-off radius of the gates. These gates are usually held to place by a lever and weight. When sufficient material has collected above the gate more than to counterbalance it, the gate opens and permits the grading to discharge. Gates of this kind are necessary to prevent the air from passing through the discharge passages, taking with it material which does not belong at these points.

Expansion chambers can be placed horizontally, on an inclination or vertically. The inclined chambers lend themselves to the simplest means for discharging the gradings as it is only necessary to suspend from the expansion chamber a series of hoppers with discharge means at the bottom, these hoppers will increase in dimensions from the entry to the discharge end. The inclined and vertical expansion chambers have the disadvantage that they take up more head room than horizontal ones. For vertical expansion chambers the simplest arrangement is a series of double cones or pyramids of increasing size, the pairs being placed in a vertical line, the smallest one being at the entry and the largest at the top. The gradings are caught within the inner cone and the air current circulates between the inner and outer cones. With the vertical forms the air has the power to balance a grain of a particular size or weight, the balance being dependent upon the velocity, and this in turn upon the initial velocity created by the fan, and the area of the cross-section at the point of balance.

Air grading is little practiced today. At one time it was a standard method for preparing sandpaper gradings. It requires large and costly settlement chambers. It is impossible to prevent eddying currents due to variations in the blast and frictional contact with the sides of the expansion chamber. It would be possible to produce uniformity in the velocity of the air currents by the use of gratings as is done in the air tunnels for aviation experimental work did not such gratings interfere with settlement.

Dust will cling to the walls of the grading chamber until the collection is so heavy it falls off in mass. These dust falls seriously contaminate the gradings. It is very difficult to feed the granulated mass evenly to the blower. The most satisfactory feeding devices are those which mechanically put the grains in motion in the same direction as the air blast and with the same velocity.

Theory of Blowing.—There is very little theory to this type of apparatus. For upward currents the balancing current from point to point can be deduced. It is of course equal to the velocity of fall the particle attains under gravity. In the inclined forms of chamber the pressure of the air current is balanced against the inclined component of gravity.[1]

[1] For particles whose fall velocity is very small owing to minute size the formula of Stokes corrected by experiment will apply. In this formula the resistance is assumed to vary directly as the velocity. Professor R. A. Millikan has long been an experimental investigator of the fall velocity of small particles falling through various media and gives as the result of his latest experimental work:

$$V = 2/9 \frac{ga^2}{\eta} (\sigma - \rho) \left\{ 1 + A' \frac{l}{a} \right\}$$

$$A' = .864 + .29\epsilon^{-1.25\frac{a}{\epsilon}}$$

g = the constant of gravity
σ = the density of air
ρ = the density of the falling droplet
ϵ = the Naperian base (2.7182818)
a = the radius of droplet.
l = the mean free path derived from

$$\eta = .3502 d\bar{c}l \text{ and } \bar{c} = \sqrt{\frac{8}{3\pi}} \sqrt{\frac{3p}{a}}$$

For horizontal expansion chambers of regular form, such as truncated pyramids or cones, the path of a particle may be calculated theoretically from the following formula:[1]

$$y = \frac{x^4 A^2 g}{8cLV^2A'^2}$$

where A is the cross-sectional area of the small end of the chamber: A' the cross-sectional area of the large end; L the length of the chamber; V the terminal velocity of the current in feet per second equal to vA/A', where v is the entry velocity of the current; g the constant of acceleration and c a factor to apply to the fall velocity in vacuo. The origin of the curve is taken through the particle at the point where it first comes under the influence of the blast.

This mode of grading is used on feathers. It is believed by the writer to be used in the dry treatment of mica. The only plant of this kind of any importance is said to use it but details of the means employed cannot be obtained. Ground mica is used in the manufacture of wall papers, lubricants, fancy paints, molded blocks for electrical insulation. About eight sizes of ground mica are made ranging from 10 mesh to bran mica.

Grading by Blowing Over Plane Surfaces.—This operation is usually a separation of coarse from fine but it might be made susceptible to obtaining more than two sizes by means for scraping up the gradings made as they are blown to their different positions. Friction is the controlling factor with this sort of a device. Since the weight of the particle increases as the cube of the diameter, but the surface exposed to the blast only as the square the finer particles are carried farther along the surface than the coarser ones. Particles of superior specific gravity are carried a less distance than ones of the same size but of less specific gravity. The operation as to a mere separation is analogous to that performed by a fanning machine for separating chaff from wheat or cereals. The final separation of black sand from gold is done by Mexicans by blowing the dry sand away from the gold contained in a batea or dry gold pan.

Grading by Projection.—Some attempt has been made to grade material by taking advantage of the greater or less distance to which particles are projected from the end of an endless belt. In using this principle advantage is taken of the fact that owing to the opposition of the air the largest particles

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In the last two formulas on p. 225 d is the density of the air at the pressure p. $\bar{c}$ is the average velocity of agitation of the air molecule. η is the coefficient of viscosity of the air at chosen temperature.

For those who wish to pursue the subject of fall velocity further the following references may be of value. The case in which the velocity is only slightly greater than the critical value, *i.e.*, that value at which the first power law of resistance begins to break down, was treated by ARNOLD in the *Philosophical Magazine* for about 1911 and 1912. There is also some experimental work on fall velocities for greater speeds in the same publication for the years 1914, 1915 and 1916. As to the time required to reach the equilibrium speed where the first power law of velocity ultimately holds, this is exceedingly simple. A particle which falls in air at a rate of say, 1 cm. a second will reach its steady rate of fall in much less than one-thousandth of a second. That problem has been worked out by Rayleigh and can be found discussed in an article of JOHN ZELENY'S in the *Physical Review* for 1910 under the heading "The Terminal Velocity of Small Spheres in Air."

[1] The horizontal velocity at any point distant x from the origin, in the expansion chamber can be obtained from the proportion $V : v_x :: x : L$ or v equals $\frac{LV}{x}$. Now by the theoretical mechanics $\frac{dx}{dt}$ equals v or $\frac{LV}{x}$ and consequently xdv equals $LVdt$. On integrating both sides of this equation there is obtained the equation $\frac{1}{2}x^2$ equals LVt or t equals $\frac{x^2}{2LV}$. In time t the particle has settled the distance $c\left(\frac{2y}{g}\right)^{\frac{1}{2}}$. On equating the two equations equal to the time t and transposing the expression above is obtained. It is assumed that the particle falls with uniformly accelerated velocity. Consult note above on this point,

should fall nearest to the pulley and as the distance from the pulley increases successively finer particles should come to rest. If the particles are of different kinds of material and with different modes of fracturing, this mode of grading may offer means for separating the constituents by screening or other ways following the action obtained by the belt travel. If the particles be of varying specific gravity their alighting position for those of the same size will be modified to some extent by this difference, the heavier particles alighting somewhat closer to the pulley: but specific gravity differences will not affect the grading so much as differences in size and shape.

C. H. Hirshfeld[1] has experimented with this mode of grading in endeavoring to discover a means of separation for various constituents of a low value ore which did not differ much from one another in point of specific gravity. He found that very high belt-speed was necessary and this required that the apparatus be massive and have good foundations. Another difficulty encountered was with the differential crawl of the material on the belt under the high speed necessary. This was overcome by having two belts in contact with one another, both running in the same direction and with the same speed, the material being conveyed by the lower one and slippage being prevented by the upper. Both belts were run horizontally.

Rolling to Eliminate Defective Shots or Other Small Roundish Objects.—This is done on an inclined flat surface the perfect and defective shot being introduced at the upper end. At the bottom there are two receptacles for receiving the two kinds of shot. The outer one, since the perfect shot roll faster and have a flatter trajectory after leaving the edge of the inclined plane, receives the perfect shot and the one nearest to the rolling surface the imperfect ones. If instead of the stationary surface a movable one were used such as an inclined endless belt gradations in perfection of sphere could be obtained.

Grading by Volume.—The first patent of record for a distinctive device of this kind is the one issued to McKesson and Rice for their Screenless Sizer (U. S. Pat. 1,044,067, Nov. 12, 1912) but earlier patents embody the principle of this one to a certain extent. The early Rittinger ore separation table which made its advent about 1853 consisted of a slightly inclined rectangular deck to which motion was given parallel to one of the sides. The motion was of a differential character and tended to advance any material fed on the deck in a direction parallel to the motion of the table but owing to the inclination the particles tended to pursue a diagonal course. The material was fed at a corner of the deck

The modern Wilfley table invented in 1895 and others of similar type which have appeared on the market since and all a tremendous improvement on the parent Rittinger patent employ a plane differentially reciprocating deck on which are mounted a plurality of parallel riffles or channels, parallel to the line of motion given the deck. These riffles taper from their greatest depth at the feed end of the deck where the mechanism producing advancing motion is placed, to the lower end where they are of but little greater than paper depth or thickness. All the machines referred to are for concentrating ore in water. On material of uniform specific gravity such as a mass of finely ground quartz the Wilfley and other tables of a similar type give a gradation from coarse at the motion mechanism end of the table to fine at the lower end.

The riffles would have very little effect on the action of the machine used in this way other than that they serve to keep the grains suspended while the grading operation is taking place. Without the riffles the water would tend to leave the mass of grains to a more or less degree leaving sticky immobile banks of material. The

[1] Correspondence, 1915.

gradation to be obtained on Wilfley and similar tables is not as sharp as is obtainable on the dry McKesson-Rice Sizer nor owing to the slight inclination is there anywhere near the capacity of the latter machine. Owing to these drawbacks riffled shaking decks have not been employed very extensively as wet grading devices but they seem

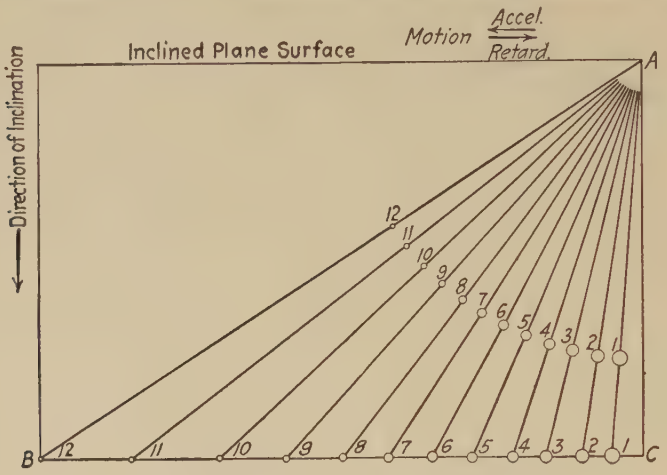

FIG. 3.—Theoretical volumetric grading.

capable of improvement and the underlying idea is suggestive. For slime and sand separation where the tonnage reaching the shaking deck is not too great, the proportion of water not too large and where both sand and slime are thoroughly in suspension the

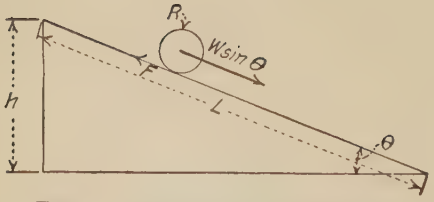

FIG. 4.—Undisturbed rolling motion.

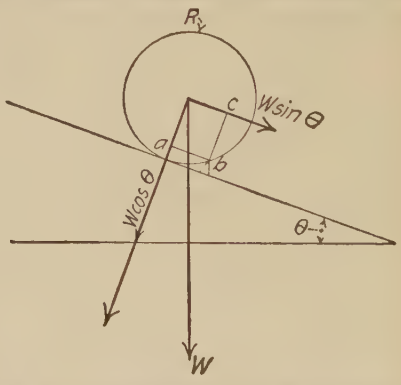

FIG. 5.—Rolling motion over an obstacle.

separation work on shaking riffled decks is excellent. By careful manipulation the quantity of water left in the sand can be reduced to a low amount.[1]

Principle of Volumetric Grading.—The essential principle governing the action in volumetric grading is the fact stated in homely language, that a large piece of rock will roll down hill faster and go farther than a small one, an experience with which every school boy is familiar. If the proper advancing motion and the

[1] U. S. Patents 741,565, 1903 and 770,877, 1904 granted to August Ten Winkel show a combination of screens, riffles and deck for improving the grading work of a Wilfley table. A more complete description of tables of the Wilfley kind appears under the head of "Concentration."

proper slope be given to the inclined deck shown in Fig. 3 the particles 1 to 12 will discharge along the edge *BC* in the order of size and position shown in the figure. This effect is obtained as must be evident by the faster rolling of the larger pieces while the longitudinal advance given the particles by the deck-motion mechanism is practically equal regardless of size. Before the particles are discharged from the side *BC* practically all diameter axes have been in contact with the grading surface, they tend therefore to report at edge *BC* according to average diameter and consequently more nearly according to volume than least size as would be the case in screening. A cube and a rectangular parallelopiped of the same volume would be expected to report together at the discharge edge in volumetric grading but could be separated by screening.[1]

McKesson-Rice Screenless Sizer.—Two types of this machine are shown in Figs. 6 and 7. Figure 6 shows a machine for making three sizes of coal from the bottom compartments and one or two more from the end *B* if this is desired and arrangements are made to that purpose. It is claimed for this machine that it can make five sizes from pieces averaging 6 in. thick down to fine slack and will handle from 40 to 80 tons an hour. Figure 7 shows a type of machine where the longitudinal advance is given by an endless travelling belt. In order to cause the particles to travel down over the fine corrugations of this machine the belt is oscillated rapidly with a device placed below it. The principal reason for adopting this type of machine on fine material is due to the difficulty of making fine dust advance under a differential shake. It has disadvantages which will be touched upon later.

Rolling friction is the chief principle of operation of the machine but to make the grading more perfect and to spread the discharge line, thus reducing the overlap at

[1] There is no satisfactory theory of rolling friction. With all surfaces the actual lifting of the rolling body over their projections and irregularities consumes the energy of the rolling body or the energy to be applied to it to cause it to roll. The difficulty of stating the problem lies in the irregularity of the opposition offered by the surface. Such opposition cannot be reduced to a mathematical equation. If the rolling opposition is negligible, which it never is, a formula for velocity may be deduced which may delight mathematicians but has no practical bearing on actualities and leads to the result that a circular ring rolls down an inclined plane of length L so as to have a terminal velocity of $\sqrt{gh}$. Down a frictionless plane it would have a terminal velocity of $\sqrt{2gh}$ and would not roll. The first formula leads to the conclusion that the velocity is not connected with the diameter which is contrary to fact.

Referring to Fig. 4 it is evident that the forces acting on the circular disc R are the gravitational component $W \sin \theta$ and the force F resisting downward motion and applied tangentially to the disc at the point where it rests on the plane. The unbalanced force is consequently $W \sin \theta - F$ and since this force acts on a mass $\frac{W}{g}$, $W \sin \theta - F$ equals $\frac{W}{g} \alpha r$, where α is the angular acceleration of the disc and r its radius. But as must be evident F also equals $\frac{W}{g} \alpha r$. On transposing, αr becomes equal to $\frac{g \sin \theta}{2}$, the linear acceleration of the periphery of the disc. Substituting this value in the general expression for velocity V equals $\sqrt{2as}$, where a is the acceleration and s the space passed, V equals $\sqrt{\frac{2g \sin \theta L}{2}}$. But h equals $L \sin \theta$, consequently V equals $\sqrt{gh}$.

The explanation given by Trautwine as to the nature of rolling friction and the factors to which it is proportional is about as satisfactory as can be obtained. His conclusions do not however give any measure of it for reasons which have already been stated. The coefficient of rolling friction has not been determined even experimentally with any degree of satisfaction.

Referring to Fig. 5, R is as before the disc and the components of the weight acting along and at right angles to the plane are $W \sin \theta$ and $W \cos \theta$ respectively. To overcome the obstruction at b there must be the relation that $W \sin \theta cb$ equals $W \cos \theta ab$. This will be evident if acb is considered a bell crank hinged at b, when to obtain a balance the moments of the two pulls must be equal. As the depth of the obstruction becomes less the leverage of $W \sin \theta$ approaches r the radius of the disc. At the same time the moment $W \cos \theta\ ab$ approaches zero. The ability of a disc to overcome rolling friction consequently increases with its diameter Also the friction is independent of the weight and specific gravity.

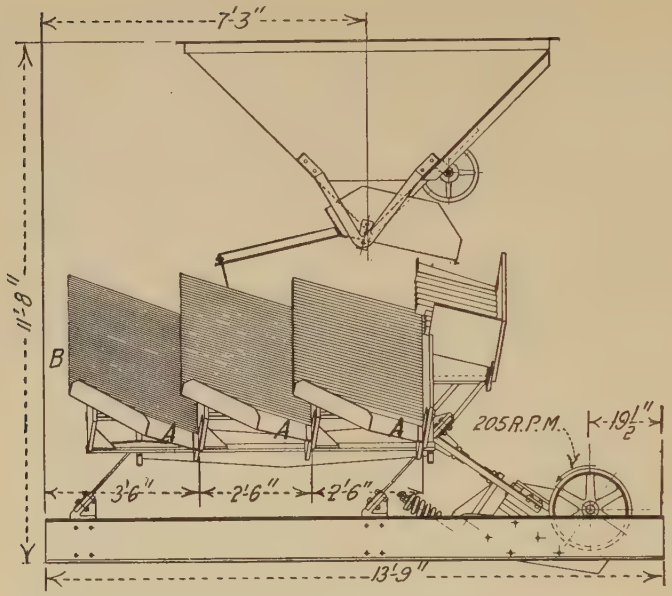

FIG. 6.—McKesson-Rice screenless sizer.

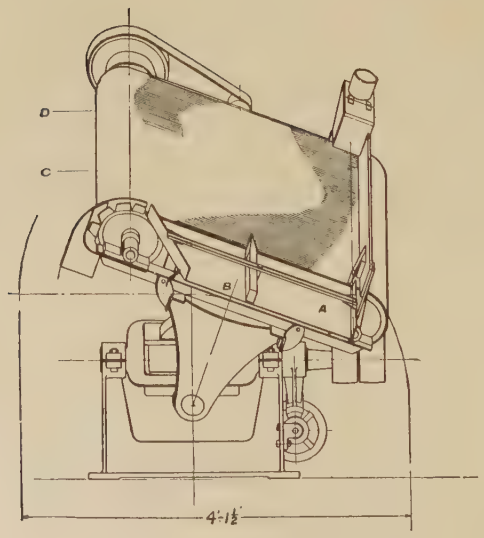

FIG. 7.—McKesson-Rice screenless sizer.

the division points the inventors modified the elementary inclined plane shaking deck. The plane surface was replaced by a corrugated one, the corrugations being parallel to a more or less degree to the line of action of the head motion. The first effect of such corrugations is to produce interstitial settlement the fines going to a point nearest the deck while the coarser particles remain at points nearer the top of the mass resting on the corrugations. By this action also the fines being more nearly in contact with the deck partake more nearly of its motion and proceed forward more rapidly than the coarse material does. The latter kind being less under the influence of the advancing motion proceeds more down the deck than along it and there is consequently better spreading and more capacity.

On the types of machine for fine grading, the larger grains in an individual corrugation, at the moment of rolling, may be considered to be resting on an inclined plane of fine material and just as on a simple plane surface they roll down faster than the finer ones do. On the machines for coarse the effect of the corrugations depends largely on whether the vertical resultant line of gravity falls inside or outside of the edge of the corrugation. If outside the particle will leave the corrugation on which it is resting for the one below. This effect is very noticeable on the coal machine. A glance at Fig. 8 will make this action clear.

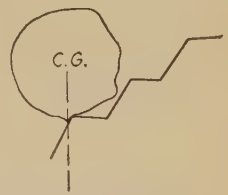

Fig. 8.—Irregular body on sizer.

In the coarse-grading machines the corrugations have a slight upward angle corresponding with the upward thrust given by the mechanism which shakes the deck. At certain intervals, depending upon the number of sizes desired, the general plane of the deck formed by the edges of the corrugations is broken into a plurality of such planes which are inclined to one another. Each set of corrugations yields a grading. The staggered arrangement of the corrugations will be plain by referring to Fig. 6.

In addition the successive sets of corrugations diminish in depth and pitch. The effect of inclining the corrugations by raising the forward ends of each set and depressing the rearward ones below the general plane of the deck, is to cause the coarser pieces to roll back, while the fines continue to advance and makes for quick separation and capacity. The effect of narrowing the corrugations is to crowd out the coarse material while the fines continue to advance and also makes for capacity. The same effect would be obtained if the corrugations were continuous and with regular inclination from end to end, but with gradually diminishing depth and pitch but such a construction would be inferior in quick separation and capacity to the broken surface arrangement. It would have the advantage that the division into sizes would not be arbitrary and could be changed at will.

On the corrugated belt machine for very fine material the advantages secured by diminishing the depth and pitch of corrugation cannot be obtained and the action of this type of machine is inferior to the others on this account. On the score of destructive vibration the belt machine is superior to the other types. On the shaking types the vibration is not only destructive to the machine but probably affects the grading work when the greatest precision obtainable is desired. In the writer's opinion balancing would overcome this defect which is a serious one but nothing has been done to redesign the shaking types with this feature in mind.

The machine has not attained a more prominent position principally owing to the reason that it has not to date duplicated the work of screens when precise work is required. This will be evident from the following screen tests.

Test on Carborundum with type-2 machine (not illustrated). Rate of feed, 15 tons in 24 hr.

Number of grade	Screens	Efficiency, per cent	Number of grade	Screens	Efficiency, per cent
1	Through 14 on 16	82.0	10	Through 36 on 50	88.5
2	Through 14 on 20	92.0	11	Through 40 on 50	77.0
3	Through 16 on 24	81.5	12	Through 44 on 58	84.5
4	Through 20 on 28	77.5	13	Through 50 on 66	84.2
5	Through 22 on 30	87.5	14	Through 58 on 74	74.0
6	Through 26 on 34	85.0	15	Through 66 on 85	78.5
7	Through 28 on 36	86.0	16	Through 82 on 97	72.0
8	Through 30 on 40	83.7	17	Through 97 on 116	73.0
9	Through 34 on 44	82.0	18	Through 97 on 139	85.5

Tests on crushed rock with type-2 machine. Rate of feeding test-*A* 380 tons in 10 hr. Test-*B*, rate of feeding 250 tons in 10 hr.

	Test *A*			Test *B*	
No. of Grade	Screen	Efficiency, per cent	No. of Grade	Screen	Efficiency, Per Cent
1	On 3/4 in.	71.0	1	On 3/4 in.	79.0
2	Through 3/4 in. on 1/2 in.	72.0	2	Through 3/4 in. on 1/2 in.	77.00
3	Through 1/2 in. on 1/4 in.	82.0	3	Through 5/8 in. on 3/8 in.	83.0
4	Through 3/8 in. on 3/16 in.	82.0	4	Through 1/2 in. on 1/4 in.	88.0
5	Through 1/4 in. on 1/8 in.	80.0	5	Through 3/8 in. on 3/16 in.	86.5
6	Through 1/8 in.	90.5	6	Through 3/16 in.	97.5

The efficiency is the percentage by weight of the proportion of the sample submitted to screen test which passes the coarser of any pair of screens and fails to pass the finer. Where only one screen is involved the efficiency is the percentage which rests upon or passes through the screen as the case may be.

It is not likely that these tests could be duplicated if the samples were fed over the machines a second time. Variations in the speed, irregularities in feeding, etc., all affect the grading work of the sizer but with improved feeding devices and more attention to the details of the machine much improvement in concordance and precision could be expected.

Screens are positive in the sense that they sharply limit the oversize of any grading they make. They are non-positive in the sense of trial for passage of the apertures. The passage of grains through or over a screening device is haphazard and even in the most precise screening many grains which would just pass through the apertures in some position do not have the chance to do so. In screening consequently there is present more or less undersize which does not belong with any particular grading.

It might be expected that the McKesson-Rice sizer would have some application in the abrasives industries but these industries have become wedded to certain fixed gradings produced by screens with sharply defined upper limits to the size of grain made in each grading. For them the McKesson-Rice machine shows too much confusion at the upper limits of the grading. Used as a rougher before finishing with screens much wear and tear of the screen cloth could be obviated.

At the lower limits the McKesson-Rice grading work also becomes confused. The machine will not separate the very finest particles. Some of the principal advantages of the device over screens are: (1) Great flexibility in changing the number and size of products. (2) Absence of the blinding and plugging found with the

use of screens. (3) Satisfactory sizing of oblong, spindle shaped or flat particles such as oats the close sizing of which is impossible on screens. (4) The elimination of the influence of specific gravity. It is possible to grade a mixture of shot of various size and cork. (5) Comminution of friable material is usually less than with screening devices.

SCREENS AND SCREENING

Theory of Screening with Flat Screens.—The splitting of loose fragmental or granular material into three or more sizes and according to one or two linear dimensions of the screen aperture is almost universally done by screens, and these devices are by far the most important ones in the whole field of grading.

If the components of a fragmental or granular mass are isometric such as are cubes, spheres, octahedrons, etc., the characteristic of a perfect or theoretical grading resulting from treatment in a battery of screens is the presence of individual particles whose axial[1] measure is but little less than the width of aperture of the two screens which take part in the grading. If the aperture be square, the axial measures of the limiting grains will be a trifle less than the linear dimensions of the sides of the two squares. If the aperture be oblong, the limiting criterion will be the width of the oblong openings. If the holes in the screens be round, the largest and smallest grains in the perfect grading will have axial measures but little less than 0.707 times the diameter of large and small openings respectively. Where the grains are spherical the limits will be but little less than the diameters.

If the grains be prisms, ellipsoids, etc., with two axes of the same length but the third longer, the criterion of the limits of the grains appearing in a particular grading will be the dimension of the short axes in the case of square and round holes, or all three of the axes if the apertures in the screen be slotted. Further descriptions of the limits of grading for regular figures might be stated, but these will suffice, for with very little extension of thought they will cover all regular figures.

If a grading be of fragmental material such as would result from crushing rock, then it is evident that if one axis is selected in a particle of such a grading, any other pair at right angles to one another and the first will within certain limits, vary as to length, the length depending upon the point they intersect the first axis. Also as the first axis is rotated in various planes its length will change within certain limits as well as the two axes at right angles to it. The limits of the length of the axes are of course the boundary of the particle wherever they happen to pierce it. What then is the criterion for the upper and lower limits of grading for material of this kind? It is impossible to state it in precise language; but quite evidently, fixing the mind on one axis as the direction axis in which a fragment is passing through an aperture of the screen controlling the upper limit of a grading, then all the other pairs at right angles to it and for any point of intersection with the first must define all planes which will be bounded by the aperture figure. Otherwise it is evident the grain will be unable to pass through. The axes of course do not, except accidentally, correspond with the extreme length of the planes which they define owing to the irregularity of the outline of the grains.

The extreme upper limit will be the dimensions of a fragment measured at right angles to the only direction axis though not to each other, except accidentally, along which the fragment can just pass through the larger aperture of the two engaged in a grading. Evidently somewhere along such a direction axis there is a plane at right angles to it which almost makes contact at three or more points with the sides of the aperture. By a similar course of reasoning the lower limit of grading for a fragmental grading can be made clear or understandable. Nothing has been said in this discussion on the screening limits of fragmental grains as to their shape. They must have sensible length,

[1] The axes are taken in the same position as mineralogical axes.

breadth and thickness. It is possible to conceive a fragmental grain having the shape of a long disk with no very sensible thickness which would just pass an aperture in a diagonal position. Grains of this shape regardless of their breadth and thickness will pass the apertures only in the direction of the longest axis. The limit for these grains is as has already been stated that they will just pass the apertures or just refuse to pass them in the case of square apertures in a diagonal position. In these cases the points of contact or near contact may for practical purposes be considered reduced to two.

Definition of Perfect Grading Device According to Diameter.—It is perfectly evident that one element of screens is positive or rigid, that is the apertures of any two screens which take part in a grading rigidly limits the over and under size. On the other hand, the trial for different axial positions is haphazard. Grains which will pass the apertures in only one or a few positions seldom get this opportunity. Fines and pieces markedly smaller than the apertures of the pair of screens with the smallest opening will be readily eliminated but not the pieces near the size of these openings. In good commercial grading a screen size will appear to consist almost entirely of grains of the size of the apertures in the screen with the smaller holes of the pair taking part in the grading.

In all cases of commercial grading the eye is in the end the sole and final criterion of grading. If the pieces comprising the grading appear to be all of a size the work is regarded as sufficiently good. The use of hand screens to test the work of power screens does little more than satisfy the eye as must be evident upon reflection. As will be shown later perfect work cannot be done by any motion imparted to a screen either by hand or power although the more prolonged the screening movements the nearer perfection and the former should be the better. Power screens are all more or less deficient in providing the complicated movements necessary for approaching nearly perfect grading. In most power-driven machines the means for progressing the material are the most important. In others where some attempt is made to duplicate the complexity of movement to be obtained by hand screening the imitation is not carried far enough or near enough to the hand movements to obtain much useful result. All such complexity also often adds to the wear and tear of the screen and the actuating mechanism. Where tonnage is the desideratum rather than perfection of screening the simplest possible progressive means usually provides for screening means which are sufficiently good provided *the screening device is not overloaded.*

In the most precise commercial grading, the best work being done in the abrasive industry, the efficiency of the work does not run much over 75 per cent. As shown by hand-screen testing the work is better than this, in some cases over 90 per cent. In making this statement about efficiency perfect work is the criterion and not the usual test work done by hand screens.

A perfect grading device according to diameter would be one which rigidly limits the size as screens do and which gives each grain a trial at the apertures in a multitude of axial presentations so as to make sure that every grain which can possibly go through the apertures does so. There is of course no screening device on the market which provides this positive action but other things being equal the nearer the screening device approaches this Ultima Thule the better it is.

Shape of Fragments.—Most material on being comminuted by crushing or shredding usually breaks up into fragments which have the same general shape in all the sizes. In the case of rock and ore crushing or other hard and resistant material the fragments are more angular in the coarser sizes and more rounded in the finer sizes. Where material breaks into fragments which to the casual

glance appear more or less cubical, scaly and longish pieces will usually be so few in comparison with the cubical shaped one that they have little or no effect on the grading work. It will be evident from what has gone before that such scaly and longish grains will in commercial work report in sizes larger than the one where they belong. In most commercial grading problems this is desirable.

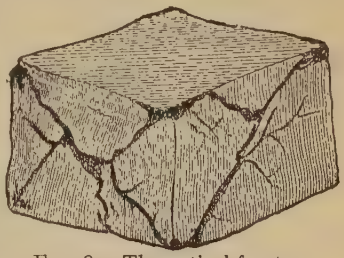

Fig. 9.—Theoretical fracture.

Material which resists crushing such as rocks and ores yield fragments which are more nearly tetrahedrons (triangular pyramids) than other shapes. The tetrahedral shape is more particularly noticeable where the rock or ore is devoid of noticeable cleavage planes. With irregular shaped fragments reëntering angles, protruding points and minor faces are noticeable but the four planes of a tetrahedron will be found to bound an irregular fragment more satisfactorily than a greater number. It is common to assume that rock fragments are more or less cubical but it is rara

Fig. 10.—Fragment of fractured cube.

to find on them six major bounding planes. The importance of this point will be brought out when capacities of screening machines of the flat type are discussed.

If a cube of isotropic material is crushed by application of uniform pressure to two parallel faces it should break up into a series of fragments bounded by conjugate planes (Figs. 9 and 10).[1] Evidently the fracturing cannot take place along the line of application nor at right angles to it and in an isotropic cube it will be at an angle of 45 deg. to the line of application of the pressure. (With cast iron the angle has been found to be 55 deg.) Since under the assumption of isotropism there can be no choice as to the number of fracture planes since all must yield simultaneously the theoretical appearance after fracturing will be as indicated in part in Figs. 9

[1] "*Encyclopedia Britannica*," XI ed.

and 10, showing the actual development of planes in a test piece. A cube fractured as in Fig. 9 will yield two pyramids, four six-faced figures with triangular faces and eight regular tetrahedrons. After the first crush of the cube the tetrahedral and pyramidal pieces in further reduction in a crushing machine will be caught between the crushing faces in such a way that a face of a fragment will be pressed against one crushing face and a point or edge of a fragment against the other. In pieces of this shape the fracturing planes will radiate from the point or edge of application of pressure. The six-sided figures having a pair of parallel faces will tend to develop conjugate planes in further crushing. In either case there is a tendency to increase in the number of tetrahedrons formed and so there is multiplication of tetrahedrons with successive comminutions.

More or less cubical blocks of large size are quite common as the ore or rock comes from the mine or quarry owing to the effect and distribution of the explosive charges in the cleavage planes. The original test cube yields 12 fragments consequently the average volume after the theoretical break is one-twelfth that of the unbroken

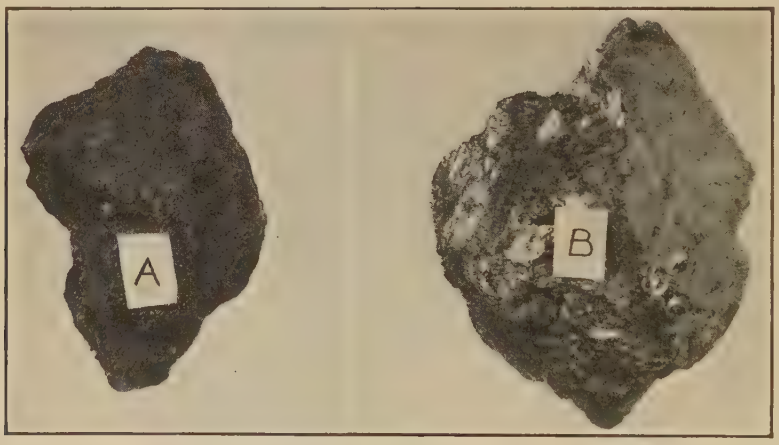

Fig. 11.—Tetrahedral ore fragments (actual size).

cube. All of these fragments require a square aperture of half the width of the original cube just to refuse to pass them or just to pass them. If the original test cube broke into others of half the length of edge of the original cube there would of course be eight of them and the volume of each would be one-eighth that of the original cube. With successive breaks there is an increase in the number of tetrahedrons formed but the discrepancy in working volume between that of a cube and a right tetrahedron of equal edge is not so great as the figures of the respective volumes would indicate. A right tetrahedron has but 11.78 per cent of the volume of a cube with the same edge. Perfect tetrahedrons with respect to the points are rarely to be found. On this account their percentage volume of the smallest enclosing cube is greater than that of any regular tetrahedron similarly enclosed. Figure 11 shows two ore fragments of the tetrahedral shape. Both reproductions are full size. By careful measurement with sliding wires fragment *A* has been found just to pass in one direction and one only a square aperture whose side is 1.2 in. The volume of the fragment is 0.6136 cu. in. obtained by water displacement. Fragment *B* will just pass an aperture of 1.5 in. and its volume is 1.4724 cu. in. The smaller fragment is consequently 35.5 per cent of the volume of the cube with side 1.2 and the volume of the larger 43.6 per cent of the volume of the cube with side of 1.5 in.

Shape of Rock Fragments in Nature.—By virtue of the pressure of mountain forming agencies more or less perfectly developed series of conjugate fracture planes are often found in nature and consequently the tetrahedral form is common. Incipient fracture planes exist in all rocks which have been subjected to pressure and with gravity are among the principal agents of erosion. Figure 12 shows a suite of the fragments of the principal rocks or rock forming minerals of a

FIG. 12.—Tetrahedral rock fragments.

Colorado mining district selected for the perfection of the tetrahedral form. All are reproduced full size. *A* is a fragment of pegmatite; *B* of vein quartz; *C* pinkish feldspar; *D* is schist and *E* granite. Where the country rock is bedded one point of the tetrahedron is often truncated and as one set of the conjugate planes will almost invariably favor the bedding planes the result of weathering of such a rock as schist is the production of fragments of flat shapes. Fragments such as *D* are difficult to find. In the district from where this came the schist is soft and most of the eroded fragments are rounded. The granite specimen came from a large boulder much broken up by fracture planes and showing tetrahedrons from 1 to 2 ft. thick to hand specimens.

Shape of Fragments as Affecting Screening Work.—Of fragments resulting from massive material those approaching roundish or other isometric shapes should screen the easiest for the reason that if they get over or into the apertures of a screen any direction axis will allow them to pass if they are undersize grains. The tetrahedral grains will give more trouble since they will usually have one position which is more stable than any other and in most forms of screening apparatus they will tend to ride through more in this position than any other. The side on which the fragments usually ride through the screening apparatus is the one of largest or nearly largest area.

Theory of Screening.—For power-driven screens the question of the degree of freedom with which grains of various size pass through the apertures is one that can best be understood from the mathematics of probability and chance. In the ensuing theory it is assumed that the fragments or grains approach the openings in a layer one grain deep and that there is no interference by other grains with a grain approaching an aperture. To put the problem in the simplest form let it be assumed, Fig. 13, that there is a square meshed opening of side l. Approaching the border of the square from any direction, and with a rate of speed some elements of which will be indicated later, is a spherical grain of diameter l/n, n being assumed to be any value equal to or greater than one. Then evidently for the grain to fall entirely within the square it must pursue some path marked by its center projected, such that at some point in its path the center projected will fall inside the concentric square $l–l/n$.

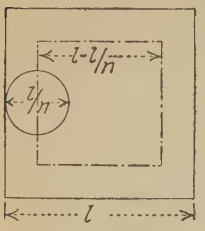

FIG. 13.

A little reflection will show that the chances of the grain falling through the square are in the ratio of the area of the inner square to the area between the inner and outer or

$$\frac{l^2 - \frac{2l^2}{n} + \frac{l^2}{n^2}}{\frac{2l^2}{n} - \frac{l^2}{n^2}} = \frac{1 - \frac{2}{n} + \frac{1}{n^2}}{\frac{2}{n} - \frac{1}{n^2}} = \frac{n^2 - 2n + 1}{2n - 1}$$

The inverse ratio fixes the chance of the grain not passing through, and is also a measure of the number of squares the average grain will have to pass over before falling through.[1]

It will readily be noted that this statement of theory places the problem in its simplest form. It would be expected that if the grain were placed in any position such that its center of gravity were within the square it would fall through the square. All such questions which by extension include impact on the sides of the aperture, the effect of which will depend on many factors, such as angle of impact, size of particle, velocity of its motion, etc., are not considered, nor, as already stated, the question of the interference of the grains with one another. Where there is screening en masse in a layer more than one grain deep, other factors being equal it must be evident efficiency of the screening must be low since the layers nearest to the screen must be disposed of before the upper ones can have trials at the apertures.[2] Where tonnage is the consideration rather than precision of work mass screening has its efficiency increased by the interstitial settlement of the fines through the bed of material on a shaking screen. This factor, helpful in giving more efficiency to the

[1] Some criticism has been made of using this ratio to express the chance. It has been said that the ratio of the inner square to the outer expresses the chance. What this last ratio gives is a measure of the percentage chance of a grain going through an aperture. Later in the discussion this ratio will be used to estimate the number of grains of a given size, out of any selected arbitrary number, which will fall through an aperture of given size. This is a slight extension of the conception under which the theory has been stated. The ratio of the inner area to the area between the inner and outer squares expresses the chance as understood in common parlance. In mundane affairs the limits of chance are infinity or nothing. We speak of the chance of an enterprise succeeding as being infinitely good or nothing. If this ratio be treason to the mathematicians let them make the most of it.

[2] "In order to appreciate the desirability of the single-particle thickness of bed in feeding screens, it is only necessary to consider the properties of the sand filter, so largely used in the cyanide process as well as other engineering undertakings, where a clear effluent is desired. A comparatively thin bed of sand will hold back appreciable quantities of slime and fine sand. In a similar way, a thick bed of feed to a screen defeats the purpose for which the screen has been designed." H. A. MEGRAW.

screen work of a more or less heavily loaded shaking screen, will be touched upon later. In screening with a single layer of particles on a flat screen the mere change of direction to which the particles are subjected by impacts does not modify the theory but helps it.

If the aperture be square and with a side of 1 in. and n be made 8, 4, 2 and 1, the following tabulation can be made of the chance of grains ⅛, ¼, ½ and 1 in. falling through the inch-square aperture and the probable number of squares such grains will have to cross before falling through an aperture of this size.

Size of Grain, Diameter, Inches	Chance of Falling in Square of In. Side	Probable Squares, 1-in. Side, to Cross, Proportional to
0	Infinity	0
⅛	3.27	0.31
¼	1.29	0.78
½	0.33	3.00
1	0	Infinity

The number of squares to cross is taken under the simplest possible assumption, *viz.*, that the probability of a grain falling through an aperture begins *de novo* at each opening and that the grains do not follow any regular path.

Capacity of Shaking Screens under the Chance Law.—If for different values of l, l/n be plotted as y and the chance as x, a series of confocal hyperbolas will result and the radius of curvature will increase as the value of l is increased. The average value of the chance for each curve, the average chance, can be obtained by the quadrature of the curve between the limits y equals 0 and l and x equals infinity and 0 and dividing the result by l. There can be no quadrature of such an area but an approximate expression can be obtained by considering the curves as equilateral hyperbolas, taking the measure for l in a unit sufficiently small so as substantially to produce curves of this kind, and then considering the curves as tangent to the axis at distances l and to equal the quarter arc of a circle of this diameter. A more exact mode of solution would be to take l/n sufficiently small for the first term plotted so that while the position of x would be at a comparatively great distance from the origin it would be at a finite distance. If this be done the value of the chance can be determined by the methods of the integral calculus.

When the average chance for different values of l is calculated it is seen at once that the average chance of grains going through the apertures of screens is directly proportional to the size of the aperture. Stated in a practical way, *in a battery of flat shaking screens other factors such as size, shape, number of shakes per minute, etc., being equal, the capacity of the screen is proportional to the size of the opening.*[1] The bigger the opening the greater the capacity and the smaller the opening the less the capacity. Screening *en masse* on flat screens or screening with screens of other types the chance law is still the governing one.

The gist of the argument on the chance law will be seen on reflecting that the more near a article is to the size of the aperture the more difficult it will be for it to pass through the screen opening and the more squares it will have to cross to be eliminated as a undersize grain. Second, since the material fed on the screens of a battery of screens consists of grains ranging in diameter from zero to a diameter but little greater than the width of the apertures, the limiting upper diameter of grain on each screen being not greater than the width of aperture of the next coarser one of the battery the ratio of the average diameter to the size of the opening becomes greater as

[1] See p. 246, *et seq.*

the size of the openings diminishes and the chance of the average grain being fed to any screen of the battery passing through its apertures consequently diminishes as the size of aperture decreases. Or in other words the capacity is proportional to the size of the aperture.[1]

So far as the ordinary shaking screen is concerned cognizance under the theory of the velocity on approach is only taken for such velocities as will allow a particle to fall within the boundaries of an aperture. The stroke of most shaking screens is of such vigor that the grains of the smaller sizes will pass over a number of apertures at too high velocities for a trial in them. The theory is only concerned with the velocities the grains have near the ends of the paths they pursue under the actuating mechanism of the screen. The ordinary shaking screen sacrifices good screen work for capacity when screening fine material.

Only the velocity case will be considered where the grain approaches an aperture at right angles to a side and along a line passing through the center of the side. The equation of a fall parabola with origin at the edge of the opening is $x^2 = -\frac{2V^2y}{g}$ where V is velocity or approach in feet per second and g the acceleration of gravity also expressed in feet. The maximum permissible velocity with the position of path assumed is obtained when x equals $l - l/2n$ and y equals l/n. Substituting these values V becomes equal to

$$(l - l/2n)\sqrt{\frac{ng}{2l}}$$

When n equals unity the conditions are obtained when the chance of the grain going through the aperture becomes 0, and V becomes equal to

$$(l - l/2)\sqrt{\frac{g}{2l}} = \frac{l}{2}\sqrt{\frac{g}{2l}} = \tfrac{1}{2}\sqrt{\frac{gl}{2}}$$

The shortest straight-line path will be one which will bring the grain tangent to two sides of the square at some point in the travel, and the longest when the grain pursues a diagonal of the square.

Interpreting the Chance Law.—In the foot note, page 238, it was stated that the ratio of the inner square of Fig. 13 to the whole square was a measure of the percentage of grains of any size smaller than an aperture which would fall through it. In the tabulation below it is assumed that there are a 100 grains each of the sizes 9, 8, 7, 6, 5, 4, 3, 2, and 1 mm. and that they are introduced to the edge of a screen of 10-mm. square openings, the stock between openings having no sensible thickness. It is reckoned under the ratio given above that in passing the first line of apertures 1 per cent of the 9-mm. grains are removed, 4 per cent of the 8-mm., 9 per cent of the 7-mm., 16 per cent of the 6-mm., 25 per cent of the 5-mm., 36 per cent of the 4-mm., 49 per cent of the 3-mm., 64 per cent of the 2-mm. and 81 per cent of the 1-mm. Traversing the second and third rows of aperture the percentages will apply to the number of grains left in passing over the preceding rows. Since the chance of 10-mm. grains passing through any number of apertures, no matter how great, is zero, this size is not tabulated. Where the calculation shows only a fraction of the volume of a whole grain eliminated it is assumed that a whole grain is eliminated where the fraction is greater than 0.5. A 10-mm. aperture is used for simplicity in computation. As an example of the computation the 9-mm. grain can be taken. Evidently in this case the area of the inner square is 1 sq. mm. and since the area of the aperture is 100 sq. mm., $\frac{1}{100}$ of the grains or 1 per cent are removed at each passage of a line of apertures.

[1] See p. 246, *et seq.*

At the first passage one grain is removed and 99 remain. At the passage of the second line of holes 0.99 of a grain is removed which is estimated as one whole grain. At the third passage 0.98 grain is removed, also reckoned as a whole one, etc.

Size of grain, millimeters	First aperture		Second aperture		Third aperture		Fourth aperture		Fifth aperture	
	Grains through	Grains remaining	Grains through	Grains remaining	Grains through	Grains remaining	Grains through	Grains remaining	Grains through	Grains remaining
9	1	99	1	98	1	97	1	96	1	95
8	4	96	4	92	4	88	4	84	3	81
7	9	91	8	83	7	76	7	69	6	61
6	16	84	13	71	11	60	10	50	8	42
5	25	75	19	56	14	42	11	31	8	23
4	36	64	23	41	15	26	9	17	6	11
3	49	51	25	26	13	13	6	7	3	4
2	64	36	23	13	8	5	3	2	1	1
1	81	19	15	4	3	1	1	0	0	0

The significance of this partial tabulation (complete tabulation for the elimination of all the grains being too lengthy) is better understood when the weight efficiency figures are computed. It is assumed that the grains are all of the same kind of material and that consequently the volume computations can be used to determine the weights and percentages. The efficiencies by weight after passing the five apertures are in round figures successively:

	Percentage
After passing first aperture.....	8
second aperture.....	14
third aperture.....	19
fourth aperture.....	23
fifth aperture.....	26

The quickness of the elimination of the small grains will be noted and also the slowness of the large grains which carry the most weight.

Commercial Screening.—The theory shows the difficulty of eliminating grains of nearly the size of the aperture and the tendency of sizes smaller than the apertures to pass into the oversizes. In the majority of the rock-crushing plants of the East and the ore-crushing plants of the West, the screening work will average to an efficiency of about 60 per cent. The various sizes produced by batteries of screens in the plants average about 40 per cent of undersize material.

In the most precise commercial grading, the best work being done in some plants in the abrasives industry, the efficiency of the screening does not actually run much over 75 per cent. As shown by hand screens the work is better than this, in some cases over 90 per cent. Hand screens as a means of testing the work of power screens are only better in degree than power screens. With hand test screens of over ½-in. size it is possible after shaking out finer particles to pick up the coarse pieces of the test sample and try them in different axial position to see if they will pass the apertures.

With fine meshed hand screens this is not possible, the eye is the sole guide and the degree of perfection of the test work will depend on the size of the test parcel and the time consumed in shaking the test screen. The theory shows that no matter how prolonged the shaking perfect results cannot be attained. It is better not to attempt to attain it. Where there is much check work to be done with hand test screens the same weight of test samples should be weighed out each time and subjected as nearly as possible to identical manipulations. This is the argument for the employment of a power-actuated battery of shaking screens.[1] The latest improvement in this device is an element for shutting off the power after the screens have been shaken any desired time.

Classification of Screens.—Mechanically screens may be divided into two classes: First, those on which the feed advances in a more or less horizontal sheet; and second, those in which a revolving motion is imparted to the material fed.

Under the first type there are the following principal kinds: (*a*) Stationary, flat and inclined, and flat and inclined shaking screens which includes shaking grizzlies of two kinds; first in which all the bars or punched, cored or drilled sheets move back and forth under the actuating mechanism as an entity; second in the independently moving bar grizzlies the bars move in parallel sets and as one set moves forward the other is moving backward, this gives an excellent transportive effect. (*b*) Endless belt or bar screens both horizontal and inclined, the first kind being the more common. In a late design of endless grizzly alternate bars are hinged and fall away when the under side of the device is reached, this detail furnishing the means of clearing the bars. When the bars return to their upper position they fall back into place. This type of screen furnishes simple transportive effect free from vibration and complicated mechanical parts but as must be evident it is deficient in means for producing good screening. Pieces of any but the largest sizes are apt on these devices to remain at the point where they are fed until they fall off at the discharge point. (*c*) Rotary flat screens, the means for advancing the material fed being centrifugal force. The surface of screens of this type is either plane or more or less concaved. (*d*) Gyratory screens. These have flat surfaces and in the flour-mill type or bolter the gyrations are in a horizontal plane. The Coxe gyrating coal screen is mounted on double cone rollers and has a rocking gyratory motion. (*e*) Batteries of corrugated cylinders mounted with their axes parallel and in a horizontal plane, have been proposed and used for grading coal or other friable material. The cylinders all rotate the same way and in a direction to advance the material fed. The undersize material falls through the spaces created by the corrugations, which match one another. (*f*) Revolving vertical cones or stationary cones fed by a revolving horizontal plate at the top.

The second type includes all kinds of revolving screens, reels, trommels, and revolving disc graders of the Rich type which have been employed on wheat, barley, malt, etc. These are usually placed with their long axis horizontal or slightly inclined. Highly inclined cones have also been used for screening of this kind. Reels much used in flour milling and operations of a similar character are run at higher speeds than trommels. Centrifugal reels are run at very high speed, centrifugal force being employed to force the flour or other fine material through the apertures of the bolting cloth. A spiral scraper or brush works the stock through the reel.

Features of Screens of the Flat Type.—If a horizontal flat screen is employed the motion imparted to it must be of a differential character, that is, one giving an accelerated forward motion and a retarded return. This sort of motion enables the grains to pick up the motion on the forward stroke and with the quick return the grains continue to advance past the return point of the complete

[1] But there is no argument for the use of washers riding on the surface of the screen.—Editor.

stroke. For fine screening and short strokes the head motions employed on ore-concentrating tables will give a satisfactory advance and the principles employed in these mechanisms can be used for greater strokes. A bumping device can also be used but has the disadvantage that it sets up destructive shocks in the screen frame and supports. If no differential motion is given a flat screen it must be fairly highly inclined or the capacity will be small. Inclinations of a few inches per foot give no results. On most material a good effect is not produced until an inclination of about 36 deg. is reached unless the motion is very strong or lively or very smooth material such as coal is being screened.

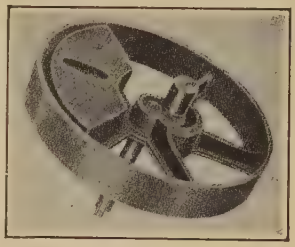

Fig. 14.—Bolter fly-wheel.

While not absolutely essential, balancing of flat shaking screens with high speeds overcomes destructive vibration. The condition for perfect counterbalance is that the momentums of the screen and counter-balance and the screen must be equal. Another mode of counterbalancing is to employ pairs of screens employed in making gradings and have the pairs move in opposition. This is commonly done with grain cleaning separators. In the Coxe and other movable bar grizzlies one set of bars moves forward as the other moves back giving a balancing effect. These bars run

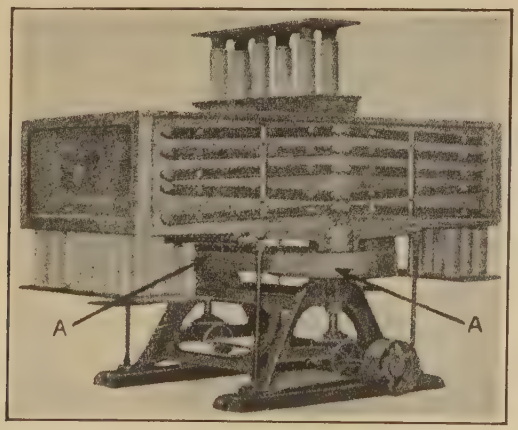

Fig. 15.—Gyratory bolter.

so slowly that balance is not needed, the transportive effect being the one sought for.

Balancing of gyratory bolters can be made quite simple. Figure 14 illustrates the flywheel producing gyration placed below one design of bolter at *A* and *A* (Fig. 15), and the means for balancing will be evident from the figures.

Interstitial Action in Mass Screening on Flat Shaking Screens.—In screening with hand screens it is noticed that if too deep a bed of material is placed on a screen, elimination of undersize is very slow. This is due to packing in the lower layers so that interstitial settlement of the fines cannot take place. With a depth of bed equal to five grains of size equal to the size of the screen aperture the packing begins to be serious. On a power screen the packing is not so bad since there is not the confinement of the material as there is in a heavily loaded hand

screen. Screening *en masse* the first principal action is an interstitial one which brings the small grains to the screen first followed by successively larger ones.

To understand this action reference should be made to Figs. 16 and 17. Figure 16 shows diagrammatically the amount of interstitial opening or amount the grains part from one another under the shaking action, *A-B* would represent the amount of opening at the top of the mass of material on the screen, while at *O* where the mass is in contact with the screen the amount the grains part from one another is zero or very nearly zero. A particle such as *G* would pass down the interstitial opening thus created to the position *C–D*, and evidently fine particles would proceed down still further, until a particle of no sensible diameter would arrive at the surface of the screen. Under the action of a shaking screen the grains tend to assume the arrangement shown in Fig. 17. To some extent this arrangement is altered by the position of the grains as they reach the screen but it tends to assume the arrangement very quickly once it is under the influence of the screening action. The shaking action actually displaces the larger grains and brings them to the top of the bed on the screen. A useful wrinkle in placing nut coal on a new or nearly dead fire, which needs coal free from fines, depends upon giving the coal scuttle a few vigorous shakes which will settle down the fines and bring the larger pieces to the top.

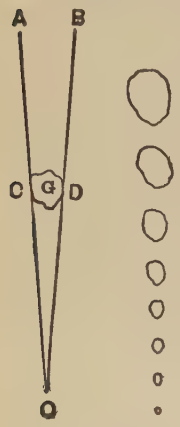

Fig.16. Fig. 17. Interstitial action.

So long as there is good interstitial effect, screening with a deep bed will yield a result sufficiently good for most commercial screening problems with non-coherent grains not too finely divided. Where extra good work is not desired, capacity will be the most important factor and as is self evident, other factors being equal the deeper the bed of material on the screen the greater the capacity. The criterion of the heaviest load a flat shaking screen could carry and give fair commercial results in precision of screening would be that at the end of the screen the layer of grains discharging would not be more than one deep. This implies the maximum amount of interference of the grains with one another which should be permitted even in ordinary commercial work. It will be understood that at entry point of the screen the material is piled up much higher than at the discharge point, there being progressive reduction in the depth of the bed as the material fed proceeds over the screen from the elimination of the undersize.

For the most precise work on flat shaking screens the bed should go on the screen in as thin a sheet as possible. The depth of bed should not be greater than the width of aperture when the material first reaches the screen. This will provide sufficient freedom of movement to the undersize grains seeking an outlet to satisfy the highest standards of screen work provided the screen is of sufficient length. In the portion of this chapter on the capacity of shaking screens to be taken up later the proper entry depth of bed for shaking screens and the proper length are translated into capacity per square foot of screen surface.

Proper Motion in Shaking Screens.—Any sort of mechanism which will give the screen a differential back and forth motion, the screen being kept in a more or less horizontal position while in motion from the actuating mechanism, will give a good screening effect. Horizontal flat shaking screens are mounted on rockers or slides or are suspended by rods. The introduction of mechanical elements to give the screen an upward stroke adds complexity and vibration, Jigging and turning over effect and aid in relieving the screen of blinding, but do not compensate for these disadvantages. The numerous patents of George

W. Cross show pierced plates with a step surface produced by bending or rolling the intent being apparently to employ such plates as stationary inclined screens and obtain a gentle rolling or turning over effect for friable material such as coal. Huge shaking riddles suspended by rods have the preference in coal breakers over revolving screens because of less tumbling about and breakage on this account.

The use of the flat or inclined shaking screen is indicated where precise work is desired, for loose dry non-coherent material not too finely divided and for high specific gravity. In grading sulphur, reels of special design are employed. Sulphur agglomerates so readily that it is frequently ruined after grading by indiscreet handling of the barrels in which it is shipped.

Gyrating screens are used on light finely divided materials such as flour stock where, owing to extreme fineness, consequent development of cohesion and adsorption, interstitial effect is *nil*, also with materials such as cork with which owing to the low specific gravity does not stratify well. On slippery and perfectly cleavable material such as mica and flake graphite gyratories find favor for each revolution of the machine tends to move the grains in a short spiral which while it spreads them well, does not afford them the opportunity of moving in erratic paths with speed too high to give much opportunity to pass through the screen openings.

In overcoming the forces of cohesion and adsorption which tend to make the grains act as a buoyed up mass it is necessary to impart a motion which will make the bed of material as thin as possible and provide numerous impacts of the grains to keep them apart and overcome the force of cohesion. On a flat horizontal gyrating screen the grains tend to move in circles. If the screen be slightly inclined the grain will tend to pursue a spiral path towards the discharge edge but its actual path is much modified by numerous collisions with other grains. Also as must be evident the long path of the stock in contact with the bolting cloth affords the time and means of breaking up its coherence and of "rubbing it through." In material such as flour stock the bulk of the grains are very finely divided and if the coherence be overcome while in contact with the screen their elimination is theoretically very simple. In bolting, blinding offers the chief obstacle to rapid screening and all gyratory bolters are provided with chains or discs or other means for overcoming blinding.

Blinding is so serious in screening damp finely divided materials such as clay that vibrating screens are preferred on this class of material to gyratory machines. On comparatively coarse loose material such as abrasives where a light feed would be desirable to secure precision of work a gyratory screen would give good results but the objection to its use on this kind of material would be the excessive wear of the screen cloth.

Wet Screening with Flat Shaking Screens.—Where manufacturing or cleaning is done in water it is often advantageous to screen in water. The principal advantageous effect of wet screening is the rapid and direct transportation of the undersize grains through the openings but as will be shown a little later this action can only be secured imperfectly. The other advantages of wet screening are the washing off of coats of dust adhering to the grains and a cheap and simple mode of transporting oversize and undersize products to and from the screens and with less head room than would be required with dry spouts.

In hand screening in a suitable vessel to hold the undersize grains which pass through the screen and the water aiding in the screen operation the thing noticed at once is that unless the mass on the screen is stirred occasionally with the fingers little or no screening action takes place. Again if the screen is pushed down too deeply in the water it is seen that the effect is not so good as where it is pushed down

to a position where the bed of particles is barely or slightly submerged. An up and down motion given to the hand screen is not so effective as the alternation of this movement with a sidewise shake. The up and down or jigging motion tends to bring the coarse particles to the screen while the fine go to the top and it is also very effective in keeping the bed loose so as to get interstitial effect while giving the screen sidewise strokes. On the down movement of the jigging stroke the bed must be submerged and on the up movement raised slightly out of the water to get the draining away which assists in carrying the undersize grains through the aperture. The side stroke to give interstitial effect must be done with the bed submerged. With power screens owing to the extra mechanical complexity added in obtaining a jigging stroke inventors have so far contented themselves with a flat shaking screen either only slightly submerged in the water or submerged during some part of the stroke. With the first arrangement waves are set up in the tank on which the screen is mounted and water gets trapped above the screen and the bed of grains becomes matted down and waterproof. This difficulty increases as the size of material fed diminishes. When the screen is only submerged during part of the stroke the matting of the bed of grains is greater. The other difficulty which occurs with wet screening is the disposal of the products made by the screen. If the undersize be allowed to flow from the bottom of the tank on which the screen is mounted the opening provided for this purpose must be sufficiently large to prevent clogging and this will require excessive amounts of water. If mechanical means are provided to remove the undersize it adds to the complexity of the design. If a flat horizontal screen be employed, it is evident that much of the water will spill over the end on the tank on which it is mounted from the wave motion set up by the action of the screen unless some special form is used. Thus J. M. Callow's patent (788,246, April 25, 1905), shows a shaking screen with a small upturned hinged portion at the discharge end, and it seems a screen with an upturned curved portion at the discharge end would give the same effect.

Wet screening with flat screens either placed horizontally or inclined and using water in the form of sprays does not give satisfaction. The water drains away very quickly, leaving a balled up sticky mass which is difficult to progress over the screen, unless an excessive amount of water is used the sprays reaching every part of the screen.

J. M. Callow has experimented with and devised a number of wet screens but the net results of his labor is the endless-belt machine with spray near the discharge end. The screen belt is mounted on the drums which support it with sufficient slack so that it droops and assumes a slight curve between the entry point of the material fed at the rear drum and the forward drum where the sprays are placed. The washing back effect at this point affords the chief opportunity for the grains to pass over the apertures. At the entry point only the water and small undersize goes through the screen the balance of the material fed matting down upon it. While the Callow screen has no movement effective in securing good and rapid movement over the screen cloth over the screen nevertheless it has attained commercial importance. The size range of the machine is from 16 to 200 mesh. Below 16 mesh the wires of the cloth are so stiff that they do not pass readily around the drums. Cloth finer than 80 mesh is usually too costly to use with this device.

Where capacity is the desideratum rather than precision of screen work the governing points are: The width of the screen and the rate at which it advances the material over it. In screens designed rather for capacity than closeness of work the load per unit width of the screen and the rate of advance of material are more important factors than the length of the screen or the area of it. For good screen work the stroke meaning here the actual distance the material advances under a pulse of the actuating mechanism the forward movements of the material fed being usually less than those of the actuating motion and tending to decrease as the load on the screen

increases, should be proportional to the size of the material fed. It should not be greater than three times the diameter of the largest grain fed though it frequently is in screening fine sizes because of practical considerations. If the stroke is too great there is apt to be no screening effect except towards the end of it when the grains are coming to rest. If a grain $\frac{1}{100}$ in. diameter is passing over a screen with $\frac{1}{50}$ in. openings and the length of stroke be 50 mesh lengths long then evidently there is little or no screening effect until towards the end of the stroke when the grain is coming to rest. If coarse crushed grains are being screened the coarsest of them being say in the neighborhood of 1 in. diameter then evidently a stroke of about $\frac{1}{50}$ in. while yielding some slight screening effect would not give little capacity. On the other hand a stroke in the neighborhood of an inch would give good screening effect and a large capacity. To equalize the capacity factor in the two cases the screen with $\frac{1}{50}$ in. opening would have to have 50 times the number of strokes that would be required for screening the coarser material. If coarse material is being screened on a shaking screen with 1-in. opening at the rate of 300 strokes per minute of 1-in. length then as must be evident other things being equal and an equal quality of screening work being desired a screen of $\frac{1}{50}$ in. must have 15,000 strokes per minute if it be desired to equalize the advance factor of capacity. It is true that if this is done that the finer grains will pass over 50 times as many apertures as the coarser ones but under the chance law the elements of which have been recited at length, the fine grains will need just this proportionately greater number of apertures to cross in order that the screen work may be as good with the fine size as the coarse.

As indicated at an earlier point the capacity of a flat screen cannot be greater than would bring a single line of contiguous oversize grains to the discharge point of the screen consequently if A is the average of the width of the two openings of the two screens involved in producing a grading, the one through which it has passed and the one over which it is passing, expressed in a fraction of a foot, then theoretically the capacity cannot exceed in cubic feet per hour per foot of screen width more than

$$\frac{5AR}{P}.$$

R being the rate of advance of material in inches per minute and P the percentage expressed decimally that the oversize bears to the whole weight of material fed to the screen and which would have to be determined experimentally by screen or other test and of which if done the first would be only a practical approximation. Printed figures on the comminution of various substances showing the proportion of oversizes to be expected from any degree of comminution can be used.

A little reflection will show that the capacity will depend upon the proportion of oversize, the greater it is the less the capacity of the screen and the greatness of the drop in size of screen opening from one screen to another. As an example of the application of the formula the capacity under it of a screen with $\frac{3}{4}$-in. opening will be considered, being fed ore or rock crushed to pass an inch-square opening. The screen is assumed to be of the flat shaking type and have imparted to it 300 differential strokes of 1 in. each per minute. Here A is the average of 1 in. and $\frac{3}{4}$ in. or expressed in a fraction of a foot, 0.073, and R, 300. Experience with rocks and ores show that the screen ratio as used in this example which is close to 1.414 the oversize of a $\frac{3}{4}$-in. screen will be about 27 per cent of the weight fed to it. The capacity of the formula under the assumptions and making the proper substitutions becomes 406 cu. ft. per hour.

Tonnage Capacity per Foot of Width, Flat Screening Surface, Rocks and Ores.—These substances in solid masses can be assumed to weigh 165 lb. per cubic foot. But as shown on pages 235 and 236 they occupy when crushed only about 40 per cent of the cube whose edge is the size of aperture through which fragments of them will just pass. The factor for multiplying cubical capacity for

conversion into hourly tonnage is consequently 0.033 or the tonnage per foot of width is 13.38 tons per hour. In round figures 13 tons.

Length of Screen.—For screening capacity breadth is called for rather than length. Greater breadth than length means increased head room for uniformly distributing the material fed across the entry end of the screen and increase of the disadvantage under which flat screens labor, the difficulty of spreading the material fed at the entry point of the screen. A length greater than 5 ft. is unnecessary and unless the screen is heavily loaded 4 ft. is ample, these lengths give the capacity on rocks and ores per square foot as 3.25 and 2.60 tons per hour respectively for material whose maximum size is 1 in.

In arriving at the practical figures for capacity extremes have been assumed. For the ordinary shaking screen 300 strokes per minute is about the limit at which the screen frame will withstand the racking actions tending to destroy it. The assumption that the terminal grains have a range only between ¾ in. and 1 in. is contrary to theory and practice. The average terminal grain would be very much smaller than this even with the most precise screening. Down to sizes of ½-in. grain and smaller and in capacity problems trommels will be found better suited than flat screens. The field of the flat screen is in the finer sizes or where friability, as in grading salt, renders the revolving screen objectionable on the score of breakage.

It will now be seen that if other factors are equal the capacities of flat screens are proportional to the size of the material fed. If the capacity of inch rock and ore material on a ¾-in. screen is assumed to be 1 ton per hour per square foot of surface then its capacity on 6-mesh screen with 0.131-in. opening after passing through a 4-mesh opening with 0.185-in. aperture, all openings being assumed to be square, is about 0.17 tons per hour per square foot of screen. The meshes and sizes of apertures are the nearest equivalent to screen openings of 0.178 and 0.125 which are the exact equivalents of the screen ratio 1.414 beginning with 1 in. With the ratio 1.414 the reduction in aperture from size to size is by halving the area of the successively diminishing apertures. The tonnage figure is computed on the exact ratio, the inch aperture is very nearly six times the width of the theoretical aperture of 0.178 and the tonnage for this size consequently one-sixth that of the larger opening.

Objections to the Diameter Rule with Diminishing Size.—The principal objection to applying the diameter rule for capacity is that if the rate of progression of the material over the screen is the same for all the sizes of screen employed in a battery of screens, and as it usually is, the screen work becomes poorer as the size of the screen and material fed becomes smaller. If the strokes of the machine are diminished as the size fed diminishes the capacity falls off.

Vibrating Screens.—A number of new models of these devices have lately appeared on the market. The vibrating element is attached directly to the screen cloth and means are provided for changing the tautness of the cloth and regulating the number of vibrations per minute. In one type of these devices 2,000 to 3,000 vibrations are given by tappets actuated through gearing. In another kind a solenoid produces the desired vibrating effect and in a third an unbalanced electric motor is fastened to the screen at either end of its rotor shaft and gives a double rotary oscillation. The mechanism produces 3,600 oscillations per minute. The chief interest in these devices lies in the attempt to correct a short stroke by a large number of them thus tending to maintain capacity while at the same time maintaining good screening effect. They should be particularly valuable for fine screening dry loose materials. It would be

expected that the chafing and bending effect on the wires of the cloth tending to wear them at the crimping points or break them by repeated flexures would increase with the coarseness of the cloth used but at this writing there is no data available on this point. All these vibrating screens are more or less dependent on gravity as a progressive force, that is they are all more or less inclined.

The rate at which the undersize passes down through the openings of screens would increasingly be diminished by friction as the openings diminish in size. If the apertures of a screen be considered as pipes for the transport of material then evidently even if the stock of which the screen is composed is reduced in thickness to a degree proportional to the width of the aperture, since the number of apertures per unit of surface increases as the reciprocal of the square of the opening, the opposition to discharge must increase as the linear number of meshes increases. As a matter of fact the ratio of the weight of stock in the screen per unit of area to the area of opening increases as the number of apertures per linear unit increases, and consequently the increase of friction as the size of the apertures is diminished is something more than can be expressed by a simple ratio.

With screen cloth the percentage of net opening diminishes with reduction in the size of opening. Some idea of this can be gotten by comparing the extremes of the Tyler sieve scale. The screen of largest opening 1.050 in. has a wire of 0.149 in diameter. The 200-mesh screen has an opening 0.0029 and a wire 0.0021 in diameter. In the first case the percentage of areal opening is about 77 per cent and in the second about 34 per cent. As is quite evident the decrease of open area with size diminishes the chances of the undersize grains passing through the apertures.

When materials are comminuted or reduced in size by crushing, grinding, shredding or other similar operations the amount of oversize tends to increase from size to size as the sizes successively diminish and as the limit of crushing is successively diminished in size. This is due to the fact that each comminution or other reducing action adds a modicum of fines and the amount of the fines becomes greater with successive reducing actions. If the limit of reduction be high and the resulting mass of particles be subjected to screening operations on a battery of screens of the same size and shape, the undersizes passing to the screen next below in size, etc., there may not be sufficient material when the finer sizes are reached to load the corresponding screens to capacity. On the other hand if the material to be screened has been reduced to a fine state of division the proportioning of screen surface according to a diameter rule based on some higher upper limit of reduction will lead to under capacity.

Cohesion and adsorption increase with reduction in size. When coarse sizes are being screened the eye will detect very quickly particles of undersize in the oversizes. With the fine sizes the eye will be ignorant of diameter variations as great 1000 per cent or even greater. In testing with hand screens with hand manipulation the individual screens are shaken until the eye is satisfied when fine sizes are being tested.[1] With material 1½in. size and larger it is a simple matter after shaking out the small undersize grains to pick up the individual pieces and try them in various axial position to see if they will pass the apertures. Below ½-in. size this procedure is very arduous. On coarse material it is possible to obtain a perfect screen test but not on fine. With fine sizes since the eye is the final criterion of commercial screening much greater capacity is satisfactorily obtained than would be permissible if exact criteria of efficiency were used.

Precise Screening.—It will be evident that with very little reduction in capacity per unit of width of screen but with increased length such length depend-

[1] The undersize can be removed and the further amount screened out in a given time noted. This is done in cement testing.—EDITOR.

ing on the degree of precision of screening required, any degree of closeness of work can be obtained. In practice, however, in those grading industries where extra high precision is demanded and obtained, as in the abrasive industries, in which flat shaking screens are much favored, the ratio of length to width of screen ranges from two to four, the higher ratios being on the narrower screens. While oblong screens are favored in industries requiring precise grading such lengths do not run to extremes for with a load not too great on the screen the slow elimination of grains of almost the size of the aperture by extra screen length does not yield any appreciable benefit. In precise grading the ratio of length of screen to width runs from two to five. From 25 to 50 per cent of the rate of feed which has been indicated as practicable for capacity screening should be employed in precise screening, experience being the guide in this factor as well as in the one of screen length.

Screen Tests.—In arriving at the number of screens of any type to be employed screen tests with hand screens should be employed. The assumption that the same number and size of screens, even when the apertures from one to another differ by a fixed ratio, should be the same for all sizes, should only be used for rough calculations. Where much information has been collected and arranged on the way any certain material groups itself into ranges of sizes following comminuting or other operations reducing it to fragments smaller than those subjected to such reducing operation, then such data may be followed as a guide in estimating the number and size of screens to be employed in any series screening operation.

For rocks and ores an approximate screen test may be written, assuming using the 1.414 ratio, by reckoning the rock or ore all crushed to pass any size of a set in which 1.414 is the multiplier and divisor and reckoning the percentage by weight which will remain on the screen of next smaller size of opening as 27, that remaining on the third as two thirds of this figure, etc., the percentage remaining on any screen being two thirds of that of next larger size and the base for the computation being 27 per cent. Following this direction if material were crushed to pass a screen with 1.050 in. opening, there would be 27 per cent remaining on the 0.742 in. size, 18 per cent on the 0.545 size, etc. If material were crushed to pass the 0.0041 in., 150-mesh size, there would be 27 per cent resting on the 0.0029 in., 200-mesh size and 73 per cent of −200-mesh size.

REVOLVING SCREENS

Revolving screens may conveniently be divided into three classes: (1) The ordinary slow-revolving designs much favored in rock, gravel and ore grading and crushing plants; (2) centrifugal reels with high speed of rotation and much used in flour milling and allied industries. Sulphur, graphite, ground drugs, pigments and other materials are also graded in centrifugal reels; (3) disc graders—in the parent or Rich machine the screen is built up of a series of annular rings or discs. The grader is bound together by a series of tie rods which pass through the inner lugs of the ring. (U. S. patent 892,825, July 7, 1908, shows a mechanical equivalent of the Rich device in a long coiled spring and by adjusting the pitch of the spring any desired amount of opening between the coils can be obtained.) The screening space is, of course, the space between the rings or the spring

coils. The rings are made this way to provide for quick discharge and reduce blinding. The outer portion of the rings is beveled and clusters of bristles fastened to a supporting arm running the length of the grader are introduced between the rings to force out any lodged material.

Material to be graded is introduced into the grader as in the ordinary revolving screen at the upper end and works its way to the lower or discharge end by gravity, the device being inclined to effect this. It will be noted that it effects a grading by one diameter or measure of volume. Its most distinctive field would seem to be for longish or spindle-shaped grains such as those of barley and wheat, which, owing to the slight difference in the short diameter cannot be graded by ordinary screens. In malting, barley is first steeped in water from 24 to 40 hr., which causes the grain to take up from 10 to 30 per cent water when it swells and begins to germinate. Following this operation the grain is piled up and the heat created by vital actions conserved, the growth being stimulated. The grain is then stirred to check the growth of the rootlets and to stimulate the growth of the acrospires. Finally the grain is dried, the germination being completely checked. A malt kiln is used for this purpose. The malt is then screened when the rootlets and acrospires fall off.

The chemical changes effected are the conversion of azotized substance into diastase, the conversion of starch into grape sugar and the imparting of color and flavor to the malt. The malt is either light or dark colored according to the amount of heat used in the kiln. When heating is prolonged more empyreumatic oil accumulates in the malt and it yields a sharper or stronger and fuller flavor. The principal advantage which would result from grading barley would be that with uniform-size berries the germination period would be about the same for all of them and the various steps in the malting process would be better defined and better regulated.

The argument for grading wheat *after* separatory cleaning, scouring and washing and other preparatory operations is that the wheat berry under the pressure of the breaking rolls breaks into flour, middlings and bran. The outer portion of the berry is a tough fibrous substance which is flattened out in the rolling and not broken into small particles as the other parts of the berry are. The inner portion breaks into flour and the layers between this and the outer branny layers break into middlings, familiar to household use as "Cream of Wheat" and under other trade names. The old millers attempted to grind as fine as possible at the first break getting as much flour as possible. In modern mills gradual reduction is practiced, the second break of the first middlings after screening yielding the best household or "first patent" flour. The argument for grading the wheat would be that the berries going to a roller mill would be of uniform size and receive a uniform grind, etc. Most millers and maltsters are skeptical as to the value of ultra refinements in practice which would be introduced by close grading of barley and wheat. Other suggested uses for the disc grader is in separating pin oats and double oats from milling oats, wheat from oats, for grading paddy after hulling to avoid the serious loss in broken rice. Possibly the chief argument against the device would be its small capacity. Where the eye is to be pleased as in obtaining uniform size nuts shelled or unshelled, graders of the disc type may find application. Spoiled nuts are now separated from heavier good ones by suction pipes.

Ordinary Revolving Screens or Trommels.—The most common type of these devices is of circular section, the screen cloth or plate being supported on spiders attached to a central shaft. This type of trommel has a *slope* of from a fraction of an inch to an inch and a half per foot, the tendency of late years being towards the higher figure of slope. Trommels are also made conical and pyramidal with horizontal supporting shafts. There is also a type which consists of a

screen cone rigidly attached at the small end to a gudgeon in the center of which is placed a small supporting shaft resting in two bearings. The screen is completely overhung, the weight being entirely taken by the two bearings. The interior of the screen is entirely free of supporting spiders and shafts and the feeding spout is carried inside the screen to the small or gudgeon end. After discharging at this point the material fed works its way through and down the screen in the usual trommel fashion.

For heavy service such as screening boulders and large pieces of rock revolving grizzlies clothed with heavy cast plates with cored screen openings and mounted on friction rollers or a combination of friction roller and gudgeon will if staunchly made give great satisfaction. Cast manganese-steel plates are the best for this type of device. Contrary to common belief friction rollers consume more power in friction than shafts and bearings. As it is desirable owing to great wear and tear not to have any spiders and shafts in devices of this kind the friction wheel mounting is compulsory but only at the receiving or feed end of the screen. At the other end the screen can be supported by a gudgeon and shaft which affords a simple way of driving it and helps to reduce the power consumed in friction.[1]

The common diameters for trommels are 36, 42 and 48 in. and the corresponding number of revolutions per minute fixed by practice is 20, 18 and 17.

Motion of a Particle in a Trommel.—The theory of the motion of the particles is imperfectly or incompletely set forth in the treatises on "Ore Dressing" but more completely in works of this kind than in other publications which have touched upon screening. In the trommel the material after being fed forms a more or less deep bank or mass occupying the whole length of the screen for a portion of the circumference near the bottom but to the side of the center line. The bank is on the side to which an arrow showing the direction of rotation would point, the rotation arrow being placed at the bottom of the screen. As the trommel rotates the bank rises and falls and at the same time the particles advance to a degree dependent upon the slope of the trommel. For a single particle the path described would be a series of curved and connected saw teeth.

A single particle of some material introduced into a cylindrical trommel in rotation, starts from the lowest position at the bottom and rises with the motion of the screen on an arc at right angles to its long axis and partakes entirely of the motion of the trommel and is in contact with it up to the point where the tangential component of its weight makes a line which coincides with the angle of friction, and then to a somewhat higher point due to the increased normally acting pressure caused by centrifugal force, which of course varies with the speed of rotation of the trommel. Above this point it will rise still higher by momentum. It then falls down toward the bottom. The rate of rotation must be very much below that which would cause the particle to cling to the screen surface and be carried completely around with it in its rotations. It is evident that the more the bank slides up and down over the screen surface the greater the opportunities for the undersize grains to pass through the apertures, on the other hand the higher the bank is carried at each swing the farther ahead the particles will fall in their sliding down paths. Richards has estimated that a particle will be carried completely around a 30-in. trommel when it has a rate of rotation of 64 times per minute. Similar figures for a 36-in. and a 48-in. trommel are respectively 58.4 and 50.6 r.p.m. The practical figures for rate of rotation which

[1] An excellent design of revolving grizzly is shown on p. 275. "The Theory and Practice of Ore Dressing," McGraw-Hill, 1915. Description of the details of trommels and their driving, housing, etc., can be found in this work and Richard's "Ore Dressing."

have already been given represent a compromise between good screen work and capacity.

Beyond the point where the particle clings to the screen under the forces which cause it to rise on the screen on the first part of its path, it rises further by momentum but it decreasingly partakes of the motion of the screen. When the particle reaches the highest point of its path it falls along an ellipse whose major axis is that of an ellipse formed by a vertical plane passed through the inner cylindrical surface of the trommel. The minor axis of this ellipse is of course equal to the internal diameter of the screen. The particle does not, however, return to a position at the bottom of the screen, but as Louis has shown, to a point above this; that is the particle will fall through an arc which is double that through which it rises from the point where it ceases to partake entirely of the up motion of the screen. On the next cycle of the up and down motion the particle will rise and partake of the entire motion of the screen through an arc which is equal to the arc through which it will rise by momentum.

The mathematical discussion of the motion of a particle in a trommel as given by Louis very well explains the characteristic swinging of a bank in a trommel. No attempt is made by him to measure the effect of centrifugal force. In the theory as stated by Richards the particle is assumed to rise to a point where its sliding angle is reached, modified by the effect of centrifugal force. This theory does not explain the swinging of the bank but it yields figures showing at what rate of rotation the particles will be carried entirely around the screen.

Rotation of the Bank.—In order to understand this the mind must be fixed on the particles in the bank collectively, remembering that each is endeavoring to pass through the cycles of motion which have already been described. The lower lines of particles in contact with the screen endeavor to push the upper ones above the point where they would be carried if alone in the screen; and it is quite evident that the moment they reach a point which is in the plane at which the size and kind of material in the trommel will slide on itself, they will slide down the face of the bank and when they reach the lower edge they will again pass up to their former position unless in the meantime eliminated by passage through the screen apertures. This action causes a rotation of the bank which is greatest at the borders and diminishes inwardly towards its center. The upper bounding surface of the bank is a more or less unstable plane. Except by pressure of the bank above or by accidental position immediately over an opening the possibility of a particle passing through the apertures is nil while the bank is rising and partaking of the motion of the screen. While the bank is ascending still further under momentum since there is relative motion between the screen and bank there is some opportunity to effect screening but as must be evident the best part of a saw-tooth path to effect screening is when the bank is sliding down. The particles which slide down the upper face of the bank come momentarily in contact with unoccupied screen surface where they may be eliminated.

The effect of the rotation of the bank is to bring the coarse particles to its periphery. As the coarse particles which appear at the upper edge under rotation will tend to roll farther and faster down the upper unstable surface of the bank and report at the lower edge and by progressive increase of this action as the material works down through the screen there is a tendency to the accumulation of the coarse particles at the bank's periphery. While the interstitial or shaking down effect is very much less in a trommel than on a shaking screen this action does exist so that the end result of the bank rotation is to bring coarse particles into contact with the screen with the interstices filled in with fine particles. It is this action which permits of

greater overloading with a trommel than with a shaking screen. Where a trommel is overloaded there is confusion of the limits of screening, so far as the eye can judge both in the under and the oversizes. The undersizes will contain particles of very nearly the size of the apertures and the oversizes will have much small particles and fines. On a shaking screen unless excessively overloaded few large particles will report in the undersizes and an inappreciable amount of fines will be left in the oversize.

Wet Screening with Trommels.—It has already been stated that the application of water to a flat screen to assist in screening is a failure, for it drains through leaving a sticky non-progressive mass behind it. Within certain limits this is not true of the trommel. The wash water for these screens is applied in a sheet on the upcoming side and while much of it falls without the trommel and does no good a certain proportion passes through the screen and flows down under the bank where it is held by the particles as it would be by a sponge and keeps the grains from cohering, lubricates the grains in their passage through the apertures and carries undersize through the opening by direct flowage. This effect cannot be obtained on a flat screen for the water from sprays drains through as fast as applied.

Effective use of water on a trommel becomes less as the size of the particles is diminished on account of coherence of the bank and ultimately the smearing over and complete blinding of the aperture unless an excessive amount of water is used giving an effect equal to suspending the material fed in water. Below ¼-in. size the application of water tends rapidly to increase the waterlogging of the bank until finally a point is reached where there is practically none of the rotation so necessary for good screening, the bank swinging up and down as a damp clinging mass.

Range of Use of Trommel.—On dry material the rotation begins to be slow and defective at about a ¼-in. size. The field of the trommel is for capacity screening between ¼ and 2½ in. Above the latter size well designed revolving grizzlies while competing on the score of wear and tear with well-designed balanced grizzlies of the type where the material fed is moved bodily along instead of being shaken along do not compete with them on the scores of first cost and capacity.

Within its field and range the trommel has the following advantages:

(1) An almost perfect balance; (2) ability to do fairly good work even when heavily overloaded; (3) It takes care of fluctuations in feeding better than any other screening device; (4) as compared with flat shaking screens there is less tendency to blind, due to the fact that there is no motion tending to jam a grain down into the apertures as there is with ordinary shaking screens. In this respect they are not superior to gyratory or flat screens which advance the material fed without any tendency to jam the particles into the apertures. The trommel can usually be easily beaten out while the screen is revolving with a piece of belting fastened to a handle. This simple method of relieving blinding cannot be practiced with ordinary types of screens; (5) with flat shaking screens of any width it is necessary in order to secure the best effect to distribute the feed evenly over the width of the screen, while with revolving screens it is merely necessary to discharge the feed into the upper end of the screen; (6) the trommel may be used as a wet screen.

The principal disadvantage of the trommel is the wear and tear on the supporting parts and the time and difficulty of making screen section changes.

Capacity of Trommels.—A disadvantage often cited against shaking screens by the unthinking is that only part of the screen is in use at any one time but on

the score of good screening the length of the path and within certain limits the slowness with which this path is traversed are the important considerations. It has already been pointed out that the more apertures the undersize grains pass over the better the chance of their elimination and again that in order to afford them the opportunity to fall through the apertures they must not proceed over the screen at too great rate of speed. It must be evident that the saw-toothed path pursued by any grain in the bank is of great length and that the grains pass over such a path very slowly. It is only necessary to exercise the imagination slightly on the great length of the saw-tooth path if it is pulled out into a straight line to appreciate this argument. On the other hand owing to the heaping up of the grains in a bank many of the grains have little or no opportunity to come in contact with the screen surface or on reaching the screen surface remain there only momentarily or but a short time. Although the path of any particle in the bank is long its path in contact with the screen may be short and haphazard or it may not reach the screen at all. No theory or mathe-

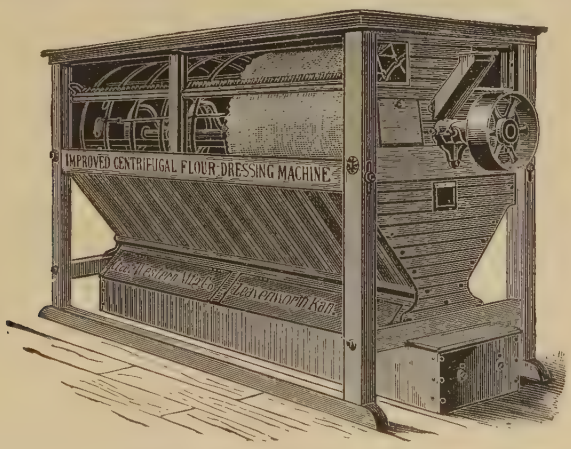

FIG. 18.—Centrifugal reel.[1]

matical analysis can properly weigh the two opposing factors of capacity in the tonnage sense and good screen work, experience must be the guide.

As a capacity screen the trommel will do as good work as a flat screen of equal width. Thus a revolving screen of 4 ft. diameter would have the capacity of a flat screen 4 ft. wide. The trommel as a capacity screen must be made longer than the flat screen but the great lengths of trommel which were formerly used are no longer in favor. As the length has been reduced the slope has been increased as it has been found this flattens the bank and gives better screening effect. As with the flat screen, for capacity screening extra length has been found to add so little extra screening effect that it does not outweigh the other disadvantages attending it. A 6-ft. length is very popular today.

Centrifugal Reels.—For light, non-abrasive, finely divided dry stock with little oversize centrifugal reels are to be recommended. These are placed with their supporting shaft horizontally or vertically and run at such a speed as to cause the material fed to them to occupy the whole surface of the screen owing

[1] Courtesy Great Western Mfg. Co.

to centrifugal force. Where screens of this type are employed, special devices must be used to hold the feed or move it in a direction contrariwise to the motion of the screen; for, as must be evident, since the grains cling to the screening surface and there would otherwise be no screening action whatever. One of the devices employed to this end is a stationary vane in contact or very nearly in contact with the screen, which holds the material fed and causes the screen to slide over it. Revolving vanes and brushes are also employed the direction of rotation being contrariwise to that of the screen.

The ordinary sizes of centrifugal reel range from 24 to 39 in. in diameter and from 5 to 8 ft. long. The 24-in. reel runs from 180 to 200 r.p.m. The reel shown in Fig. 18 runs from 100 to 130 r.p.m. and is made in diameters ranging from 24 to 30 in. The figure shows a vane revolving contrariwise to the motion of the screen.

Blinding of Screens and Methods of Prevention.—Blinding is of two kinds: (1) agglutinative, as with flour mill products, which is relieved by brushing or rubbing on yielding screens; and (2) of the sort where the particles are not de-

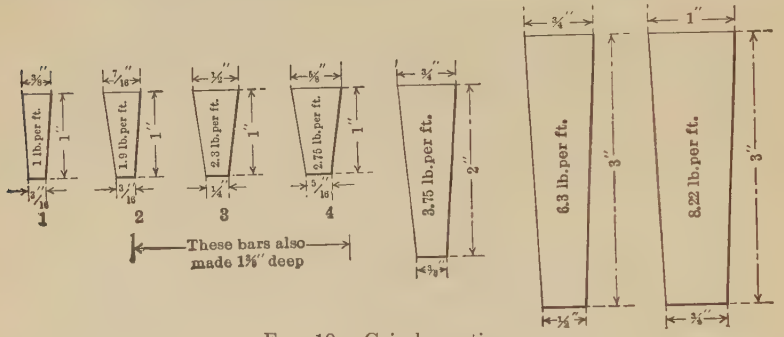

Fig. 19.—Grizzly sections.

formed but are jammed in the apertures. Square apertures are worse than round in this respect. In flour bolting machinery, loose discs of various shapes are used, being allowed to slide around freely on the bolting cloth. Chains also are used for the same purpose. For revolving screens, particularly those of polygonal shape which lend themselves to a ready adaptation, tappet devices are employed; but these are not very helpful and have the disadvantage of loosening bolts and other parts to the screen. For ores and rocks the occasional beating out of the revolving screen with a piece of belting mounted on a handle is probably about as efficacious a method of clearing the apertures as is practicable. At one of the Western ore mills relief from excessive blinding of a fine screen was obtained by mounting the trommel on a sufficiently light shaft, so that this member and the screen, by suitable means, could be kept in a state of rapid vibration.

For flat screens, rope beaters with a number of tappets have been proposed, one end of the rope being secured to the discharge end of the screen and the other fixed to the frame of the actuating mechanism. Traveling brushes and scrapers are also employed. (See Figs. 1 and 2 in the next section.) When the rope with its attached tappets is drawn taut by the advance of the screen, the tappets give the under side a sharp blow. Endless-belt screens may possibly get relief from blinding to some extent by the wedged grains being forced out on passing over rollers. Roller systems

also have been proposed for revolving screens. For medium soft materials these devices may give some relief, but where the wear on the screen is rapid they would interfere with patching.

SILK CLOTH AND GAUZE SCREEN

Number of cloth	Mesh		Gauze			
	XX cloth	XXX cloth	Number, gritz gauze	Equals cloth number	Number XXX gritz	Equals standard gritz
0000	18	...	14	....	14	16
000	23	...	16	0000	16	18
00	29	...	18	....	18	20
0	38	...	20	....	20	22
1	48	...	22	000	22	24
2	54	...	24	....	24	26
3	58	...	26	....	26	28
4	62	...	28	00	28	30
5	66	...	30	....	30	34
6	74	...	32	....	32	36
7	82	74	34	....	34	38
8	86	82	36	....	36	40
9	97	86	38	....	38	42
10	109	97	40	0	40	44
11	116	109	42	....	42	46
12	125	116	44	....	44	48
13	129	125	46	1	46	50
14	139	129	48	....	48	52
15	150	139	50	2	50	54
16	157	150	52	....	52	56
17	163	157	54	3	54	58
18	166	163	56	....	56	60
19	173	...	58	4	58	62
20	178	...	60	....	60	64
21	182	...	62	....	62	66
25	200	...	64	5	64	68
			66	....	66	70
			68	....	68	72
			70	....	70	
			72	6	72	
			80	7		
			86	8		

NOTE.—Cloth and gauze 40 in. wide. Cloth in four weights: standard, X, XX and XXX. Mesh is exact count per linear inch.

Screen-section Materials.—The materials most commonly used for making screen sections are: (1) Punched, drilled and cast plate with cored holes; (2) grizzly bars, some of the more common of the weights and dimensions of which are shown in Fig.

19; (3) brass, steel and cloth of other metals and alloys. Iron and steel cloth is for special purposes and can be tinned, galvanized or sherardized. (4) Silk screen or bolting cloth and silk gritz gauzes. Gritz gauzes are made in three weights of thread—X, XX, and XXX. Metal-screen cloth can now be bought with openings up to 4 in. and with wires up to 1 in. in diameter. At the other extreme a brass-screen

Opening in inches, ratio $\sqrt{2}$ or 1.414	Opening in millimeters	Mesh	Diameter of wire, decimal of an inch
1.050	26.67	...	0.149
0.742	18.85	...	0.135
0.525	13.33	...	0.105
0.371	9.423	...	0.092
0.263	6.680	3	0.070
0.185	4.699	4	0.065
0.131	3.327	6	0.036
0.093	2.362	8	0.032
0.065	1.651	10	0.035
0.046	1.168	14	0.025
0.0328	0.833	20	0.0172
0.0232	0.589	28	0.0125
0.0164	0.417	35	0.0122
0.0116	0.295	48	0.0092
0.0082	0.208	65	0.0072
0.0058	0.147	100	0.0042
0.0041	0.104	150	0.0026
0.0029	0.074	200	0.0021

For Closer Sizing 65 to 200 Mesh

Opening in inches, ratio $\sqrt[4]{2}$, or 1.189	Openings in millimeters	Mesh	Diameter of wire, decimal of an inch
0.0082	0.208	65	0.0072
0.0069	0.175	80	0.0056
0.0058	0.147	100	0.0042
0.0049	0.124	115	0.0038
0.0041	0.104	150	0.0026
0.0035	0.088	170	0.0024
0.0029	0.074	200	0.0021

cloth may be purchased with 300 meshes to the linear inch, the diameter of the wire used in this screen being 0.0016 in. and the net linear opening 0.0017 in. Owing to the confusion in the gages under which wire cloth is sold, one principal manufacturer now states his catalog list in mesh but the varying openings and sizes of wire are given in decimal fractions of an inch under lateral columns.

For ordinary revolving screens punched plate will be found better than cloth down to about ½-in. openings. The percentage of opening of the punched plate is less than with the cloth but the plate is more easily attached to and held on the screen supports. Silk cloth is preferred to metal even in some industries where metal cloth would at first thought seem to have the preference. One very prominent manufacturer of abrasives of great hardness prefers silk to brass cloth because it does not blind so readily. Moisture and sudden temperatures are destructive to silk cloth, the first because cloth and gauzes are held together with size. A sudden lowering of temperature will often cause a tightly stretched silk cloth to tear. An accompanying table gives the meshes of cloth and gauzes and also the equivalency in opening between cloth and X gritz gauze, and X and XXX gritz gauze.

Screen Ratio for Testing Screens.—The advantage of the screen ratio 1.414 is stated in the words of the following quotation concerning it from a W. S. Tyler trade publication: "The ratio between the different sizes of the screen scale has been taken as 1.414 or the square root of 2, as recommended by Rittinger in his work on ore dressing. The niceness of this will be apparent from the following: taking 0.0029-in. or 0.074 mm., the opening in the 200-mesh sieve as the base or starting point, the diameter of each successive opening is exactly 1.414 times the opening in the previous sieve. It also makes the area or surface of each successive opening in the scale just double that of the next finer or half that of the sizes have a constant ratio of 1.414 while the areas of the successive openings have a constant ratio of 2.

"Another advantage in this selection of ratio is that by skipping every other screen you have a ratio of diameter of 2:1, by skipping two sizes you have a ratio of 3:1 (approximately), and by skipping three sizes, you get a ratio of 4:1." The equivalent for the ratio in the ordinary range of screen cloths for testing is shown by the accompanying table. These are all standard Tyler screen sizes.

Where a closer sizing is required in the finer openings, a scale is shown below this table from 65 to 200 mesh, in which the openings increase in the ratio of the fourth root of 2 or 1.189.

SECTION VII

MECHANICAL SEPARATION

By Edward S. Wiard[1]

Introduction.—This subdivision will fall into the following sub-heads: Separation of solids from solids; separation of solids from liquids; liquids from liquids; and the separation of solids from gases. The separation of liquids from liquids is also covered under "Distillation."

Separation of Solids from Solids.—Since the preceding section dealt with screening it is natural to begin this section with separating devices employing screens or apertures. With respect to separation by an aperture there are numerous devices employing a fixed aperture where the material to be separated has a hard and soft part as cotton bolls, raisins, cherries, etc. In the case of the separation of cotton from the boll the description of the earliest gin invented by Eli Whitney will show how an aperture or apertures is made use of to free the cotton from the seed. The fiber rests upon an inclined grid into the apertures of which project a series of saw-toothed discs mounted upon a revolving shaft. The saw-toothed discs catch the cotton and pull it away from the seed and since the grid apertures are smaller than the seed the latter cannot pass through them and when clean make their way out of the device by rolling down the inclined surface of the grid.

Raisin Seeder.—The elements for one device of this kind are a cylinder mounted with closely spaced pins placed radially. The spacing of the pins is smaller than any thickness of the seed. The other elements of the seeder are a hopper in contact with the pin mounted cylinder, means for squeezing the raisin into the pins of the cylinder as by a flexible or flexibly mounted roller or rollers. Since the pins are so closely mounted that the seed cannot be squeezed in between them only the meat of the raisin is pushed into the interstices and the seeds project above the pin cylinder. Means are provided for removing the seed as by a scraper or toothed roller and finally means for removing the meats by a saw-toothed scraper. The operation of the household cherry stoner is too familiar to need description.

Separation by Screening.—A description of the mode of handling wheat in a large flour mill for cleaning and separating will illustrate the possibilities of separation by screens. Since work of this kind is done on a type of screen known as a separator a description of this machine is necessary for understanding the the ensuing account of the separating operations. Figure 1 shows a common type of separator, the wheat entering the top and first passing under the winnowing influence of the fan shown at the top of the figure. From this point it proceeds downward on to the screens. In the separator of the figure there are two batteries of screens, three high, which oscillate in opposition to one another so as to produce balance and eliminate vibration. The pair of actuating eccentrics are shown at *A* and *B*. The two batteries of screens slope in opposite directions. To overcome

[1] Consulting Engineer, 409 Boston Building, Denver, Colo.

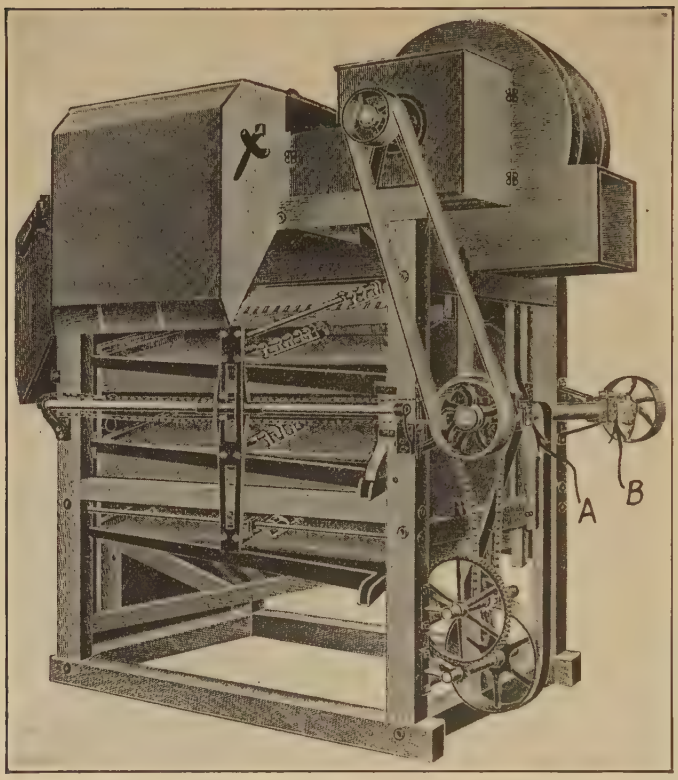

FIG. 1.—Wheat-grading and cleaning machine.

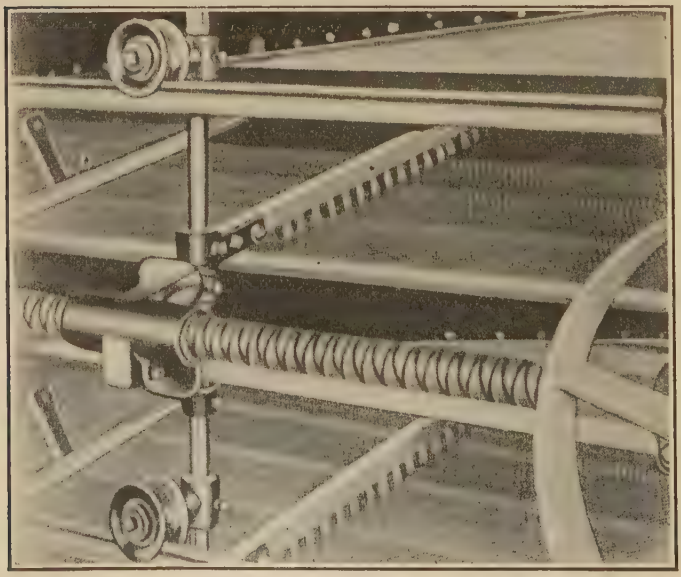

FIG. 2.—Traveling brushes to overcome blinding.

blinding either rakes traveling back and forth in contact with the screen are used as is illustrated in Fig. 1 or brushes, see Fig. 2. Figure 2 shows a double-threaded screw for actuating the brushes and reversing the direction of travel at the ends of the screens. Punched steel plates are used for screens on the separators.

The wheat first goes to a receiving or scalping separator with large screen holes, ⅜ and ½ in., the oversize from this screening operation yielding all sorts of field refuse, coils of fibers, twigs, stones, bits of coal, etc., which go to the boilers as fuel. The undersize or partially cleaned wheat then passes to a separator with ¼-in. holes for the first separation, yielding wheat, which fails to pass these holes or passes them but slightly and undersize material consisting of broken wheat or small seeds or small grains which is ground for stock feed. The material freed from this small stuff passes to screens on the same frames with 3/16-in. holes, the wheat passing through and the oversize consisting largely of oats and corn goes to a special separator to be further separated for stock feed. In the mill where the particular separation means being described are used, the upper section of each screen of the No. 2 separator is 10 in. wide, the lower section with its 3/16-in. holes being 14 in. wide. Considerable variations in the separatory practice exist in different mills.

The special separator for the oats-corn mixture is a single two-high machine. The upper screen has ⅜-in. holes. The oversize of this machine is refuse. The screen below on to which the undersize of the upper discharges has 3/16-in. holes. The oversize is a corn product. The undersize is mostly oats.

The wheat leaving the 3/16-in. screen of the No. 2 separator undergoes scouring, washing and other conditioning operations following which it again passes through separators to remove sand, bran, etc., before undergoing the first rolling operation.

Cockle Cylinders.—It seems proper at this point to describe the principle of separation involved in the cockle cylinder since these devices are employed in flour mills and grain elevators in conjunction with separators of the kind already described or stationary screens of the needle-slot kind. Cockle is a generic term for small weed seed.

The cockle cylinder has on its interior numerous and closely spaced indentations. If the intent is to separate wheat from small weed seed the indentations are made just large enough to receive the weed seed but not the wheat. The cylinder is inclined and it revolves, the progress of the wheat is consequently the same as in a trommel. The weed seeds in the depressions are carried around by the revolution of the cylinder and fall out by gravity into a fixed trough which runs through the length of the cylinder. This trough is provided with a spiral conveyor for carrying off the weed seed. By varying the size and shape of the indentations other separations can be effected. These devices have been developed in the grain business but they or the principle involved have application outside of it. The capacity of cockle cylinders ranges from 20 to 25 bu. per hour. The diameter ranges from 20 to 28 in. and the length from 7 to 11 ft.

Air Separation.—The separation of fine material from coarse by bringing a stream of material to be separated under the suction of a fan has been mentioned in describing the grain separators. The means for separating the solid particles from the air will be described later. Conical separators are much used for this purpose and in grinding and separating systems such as those of the Raymond Brothers impact pulverizer the grinding and separation are done in close circuit. The ground product is aspirated into a cone the coarse falling back into the grinding machine. The fine product which fails to fall back is collected in a centrifugal cone and is removed through a gate from time to time. The air from the second

cone returns to the mill. The Raymond or similar processes is adapted to the grinding of phosphate rock, pigments, amorphous graphite, talc, soapstone, fuller's earth, bone black, shellac, bark, roots, herbs and drugs, bone, etc.

Air separation can be done by either blowing on the material to be separated or by suction. Separation by aspiration is usually to be preferred to directing the blast against the material to be separated as this mode tends to disturb the larger particles and even to carry them up into the air current.

In the separation of asbestos from gangue by aspiration methods the fluffy particles of asbestos are the ones taken up by the air current, the sand and refuse falling back in the separating devices and either going to waste or being further treated to yield inferior grades of asbestos.

Separation with Air as an Adjunct.—Two devices for separating by the aid of air currents passed up through a bed or mass of material will illustrate the principles on which these machines operate. These machines should be described under the head of "Concentration" as they employ principles of separation which have received their greatest development in that art. But since in the solution of some problems for which they are advocated there are but slight differences of specific gravity in the parts to be separated it is thought best to treat of them here.

Middlings Purifier.—For the purposes of mechanical separation the wheat berry may be considered to consist of a center of starchy material or white flour and a coat wrapping this and containing a larger proportion of gluten, which substance being absent in the loaf there results heaviness and sogginess. The deficiency of gluten in barley renders it unfit for bread making. There is finally bran surrounding the gluten coat. After wheat receives its first rolling there results much of the weak flour from the central portion of the berry, some grains of the coat of gluten and some gluten grains with bran attached and finally, bran. The weak flour is readily removed by sifting. The oversize of the sifting or middlings contains all the parts of the berry which will make the strongest and best flour provided the bran can be removed. The problem of purifying the middlings was solved in the 60's of the last century by the middlings purifier invented by LaCroix, Smith and others, the appearance of the purifier being followed by much bitter litigation and strife over the priority of the invention. Smith's purifier which has existed with little change since his time consists of a flat shaking screen in an enclosed box.

The middlings are fed in at one end of the screen by means of a roller feeder so that there will be no escape of air at that point. The long relatively narrow screen is actuated by an eccentric outside the box and the middlings are progressed by the differential secured by inclining the rock legs which support the screen. During the progression along the screen the bed of middlings is subjected to an air current which passes up through them and the screen, this current being created by a fan placed in the top of the box. Mounted on the screen frame are sections of screen cloth the meshes of which sections become progressively coarse from the feed end to the lower end of the screen frame. The enclosing box is also divided internally by baffles into compartments so that there may be individual regulation of the air currents for each screen size. To regulate the air currents gates are placed in each compartment at either the top or bottom of the enclosing box. Originally the regulating gates were placed in the top of the device but the bottom is now the preferred position. If the gates be placed in the top of the box there is danger of material adhering to the gate passages and falling back on the screen. The single fan draws the air up through

all the compartments and if one gate is given greater closure the strength of the currents increase in the others.

In the first compartment a feeble current is maintained just enough to float up the lighter impurities but not sufficient to prevent the small fine purified middlings from passing through the screen. The strength of the air current is progressively increased in the other compartments and by thus using graduated air currents the loss of middlings which would follow with an air current strong enough to remove all the impurit es is avoided. The device is provided with a brush for unblinding the screen cloth which moves in contact with the lower side of the screen cloth and is actuated by an endless chain. The purified middlings passing through the screens are removed by a worm in the bottom of the box. The bran and impure middling removed by the fan are recovered by cloth filtration. The capacity of these devices ranges from 6 to 30 bushels per hour. The area of screen cloth ranges from 7 to 20 sq. ft. The overall dimensions of the enclosing box or case range from 6 ft. height by 7 ft. length and 4 ft. width to 8 ft. height, 11 ft. length and 5 ft. width. The screens are shaken between 500 and 600 times per minute.

Mechanical arrangements similar to the middlings purifier have been employed on separation problems in a number of industries. They have met with some success, for example, in separating the woody part of the cork bark from the pithy.

Sutton, Steele & Steele Table.—Some of the separating problems involving slight differences of specific gravity or requiring merely a grading to effect the separation, for which this device has been advocated are: Grading of walnuts in the shell, peanuts in the shell and shelled and seeds of all kinds. The inventors state that in the grading of these they have noticed that the discard contains the most of the weevil germ. It is also recommended for cleaning cereals and rice and for separating cork. The machine was originally developed as a dry-ore concentrator and its arrangements resemble that of a Wilfley concentrating table. They consist of a flat deck mounted a short distance above the floor level by means of rocking legs. The deck is clothed with cloth and for treating coarse materials more open cloth is used than for fine materials.

A reciprocating motion is given to the deck by an ordinary eccentric and since the rocking legs incline towards the eccentric or head motion end of the machine, at each forward stroke the deck rises slightly and falls back an equal amount on the return giving the necessary differential for advancing feed along the deck. An ordinary centrifugal fan is built into the frame of the machine and a flexible connection leads to an air chest under the cloth mounted deck. Regulation of the air through the cloth is obtained by a gate on the suction side of the fan. A series of shallow wooden riffle strips are secured to the deck parallel to the line of the pulsions. Material is fed at the upper right-hand corner looking down the deck from the head-motion end. It spreads out over the deck under the combined influence of the advancing motion and the slightly transverse inclination given the deck. By interstitial action due to the shaking of the deck, explained elsewhere in this work, the small particles work down through the bed of material on the deck and advance to the lower end where they are discharged. The large particles which stay on top of the bed move under the influence of the transverse slope of the deck more nearly across the table and discharge over the long side. The deck is 10 ft. long and 5 ft. wide.

The riffles assist in guiding the small particles to the end of the deck or what is the same thing prevent them from working transversely across the deck with the large particle. The riffles are of such height that they do not interfere very much with the transverse travel of the large particles. If the small particles are of greater specific gravity than the large there will be separation according to gravity. If the

fragments or particles are all of the same composition the machine will effect a grading. The function of the air blown through the cloth is to keep the bed loose so that there will be no packing thus permitting interstitial action to have full play. It has been stated that the air actually raises and holds in a state of balance the larger particles. It is true that with fine materials the air creates so much looseness in the bed that the upper layers appear to be in a state of suspension and with closely sized material such a conception of the principle of operation may have some modicum of truth in it.

But it is really a superficial and unnecessary notion of the principle of operation. In the case of the middlings purifiers there is actual raising of some of the small light particles from the lower portions of the bed to the top by aid of the air current but by far the larger part of the light material is brought to the surface of the bed by interstitial action and is aspirated out of the machine from this point by the action of the fan. Aspiration is not an effect desired on the Sutton-Steele table. It will be evident on consulting the concentration section of this work that since large particles tend in a fluid or current to fall faster than small the governing principle of separation cannot be an air current. In the next described machine suitable for material of a non-packing character, separation into layers of different sized particles takes place entirely by interstitial action and without aid of air currents at all.

Separation by Interstitial Action and Deflection.—Machines of this type are largely used in oatmeal factories for separating the groats (hulled oats) from the unhulled oats. They are also recommended for separating hulled from unhulled or paddy rice, garlic from wheat, and stone from wheat. The last application seems somewhat dubious. The apparatus consists of a shaking frame on which is mounted one or more flat decks. An actuating mechanism shakes the decks back and forth and there are means for changing the slope of the decks and a speed-changing device so that the number of throws per minute can be altered while the machine is running. The decks are separated into 6 to 12 compartments each facing openings in the feed box placed along the upper sides of the decks. The compartments on the decks are defined by triangular deflectors of steel plate the sides of the deflectors being at right angles to the plane of the deck. On groats and hulled oats being fed to the decks the greater slipperiness of the latter and their smaller size allows them to settle through the bed of mixed material on the decks. Owing to the transverse slope of the decks the lower layers consisting largely of the hulled oats will work down to the lower side of the deck where they are discharged. The groats in the upper layers partake more nearly of the back and forth motion of the decks and meeting the deflectors are carried up the deck contrariwise to the direction of travel of the hulled oats. The groats travel toward the upper or feed side of the machine and discharge over a dam or baffle placed along that side of the decks.

In the case of separation of garlic from wheat the garlic would follow the line of travel of the groats because of its greater size. It is probable that in all cases of separation on this device slight differences in density assist the separatory actions (see page 243 *et seq.*).

Separation Depending upon Sliding Friction.—Two devices both from the anthracite-coal industry will illustrate the application of this principle of separation. According to Ayres, the inventor of the separator shown in Fig. 3, anthracite coal may be grouped into the following classes:

1. Glassy fracture coal, usually cubical in form.
2. Flat coal, some pieces having slate faces.

3. Bone (interlaminated coal and slate), usually flat and either coal-faced or slate-faced.

4. Flat slate from 0.25 to 0.5 in. thick.

5. Pure slate with coal faces, approximately cubical in form.

6. Slate and rock, heavy, and cubical in form.

Of these classes (2) and (5) cannot be separated by a fractional separating device. In separating by specific gravity fairly good results are obtained in separating groups (2) and (5) but in using jigs, the most common sort of gravitational separation device, much glassy fractured coal, class (1) goes over into the slate discharges with classes

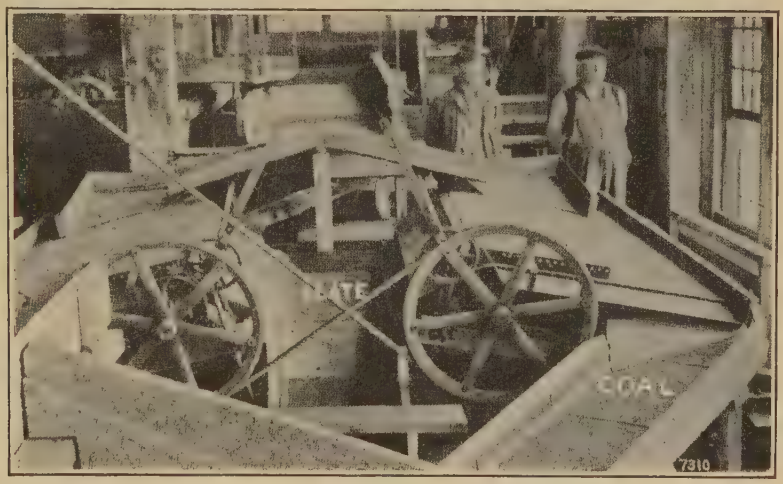

FIG. 3.—Separation by friction.

(5) and (6). The point these observations bring out are that none of the separation devices for coal are perfect and that even to get good commercial results additional separation means must be used. In the case of the separator under discussion the work performed is of a roughing character and the inventor advocates hand separation or jigging or both following its use.

Figure 3 shows the general arrangement of the machine. The traveling separating belt is mounted on two shafts and the direction of travel of the belt is upward towards the material receiving side. The traveling belt is inclined forward as well as downward in the direction of travel. The material is fed in a continuous stream at the farther left-hand corner and as it slides off the feeding pan it is immediately spread out on the separating belt. The coal slides down until it comes in contact with the guide at the lower end and then works its way along the guide to the discharge point at the lower right-hand corner. The slate is carried up and discharges at the upper end.

The capacity of the machine ranges from 5 tons an hour on pea size (through 0.75-in. and over a 0.5-in. screen) to 25 tons an hour for steamboat size (through a 6-in. and over a 4.5-in. screen). On steamboat size four of these separators and 12 men do the work that 32 men do by hand sorting.

For separating coal from narrow seams the complete process of separation would involve an arrangement as follows: First the use of the Ayres separators which will divert from 60 to 75 per cent of the total pure coal when properly adjusted. The impure product is run over a second set of separators to remove all the flat slate, class (4) and all pure slate and rock, group (6). The reject from the second set of

separators is then jigged principally for the object of separating class (2), flat coal with slate faces from classes (3) and (5), bone and pure slate with coal faces. The gains by this mode of operating over the old way by straight jigging yield an increased gain in coal up to nearly 20 per cent.

Spiral Anthracite Separator.—Figure 4 shows the centrifugal spiral separator for anthracite coal. In this design there are three receiving divisions or chutes at the top. By centrifugal force and greater ease in sliding the clean coal rides to the upper edge of the inner spirals and drops into the large outer spirals, being discharged at the bottom. The inner or slate spirals can be arranged so as to give two products. The surface of the slate spirals are arranged with deflecting elements to assist in the separation. A capacity from 5 to 10 tons an hour is claimed for these devices.

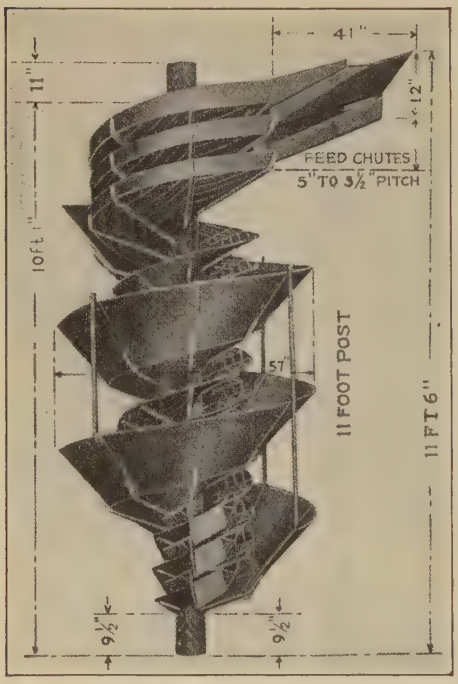

FIG. 4.—Centrifugal separator for anthracite.

Separation by Decrepitation Followed by Screening.—Minerals like calcite, fluorite and barite which decrepitate on heating may be separated from gangue by roasting followed by screening.

Decortication and Separation of Fibers.—By decortication is meant the removal of a coat and since in the preparation of vegetable fibers the cortex must be removed at some stage of the operation, this term has taken on the additional definition of mechanically cleaning fibers.

The cortex of flax is its valuable part and consists of a layer of fibers surrounding a woody core or center. Ramie consists of bundles of fibers held together by a very sticky gum and enclosed by cortex. In preparing flax it is passed through a series of corrugated or toothed rolls, the corrugations or teeth meshing more or less with one another. This breaks up the woody core without materially injuring the fibers and upon passing the flax through a threshing or scutching machine the woody core is removed. Preparatory to pasage through the breaking rolls the flax undergoes a fermentation process to open up the bundles of fibers composing an outer layer of the stems.

The cortex of ramie can, though without much success, be removed by passing it through a beating engine with revolving beater arms. The ramie comes under the influence of the beaters as it passes over a fixed shoe. The action of the machine loosens some of the sticky binding gum but not all of it. Ramie while it is a very strong fiber, the strongest there is in tension, tends to break if bent sharply and the

beating engine has consequently a bad effect upon the fiber. The gum can be removed if the plants are steeped immediately after cutting them down. After the gum has become hardened caustic soda is required to remove it, and this chemical has an injurious effect on the fiber.

In the decortication of grains the cortex or bran can be removed by abrasion. The devices used for this purpose are commonly double roughened cones, the arrangement of cones being about the same as in an ordinary coffee grinder. Pearl barley is prepared by machines of this kind. It has been stated that the Romans were familiar with the decortication of grain by grinding it in a mortar with abrasive grains and the coats of any cereal can be removed by mill stones, the upper stone

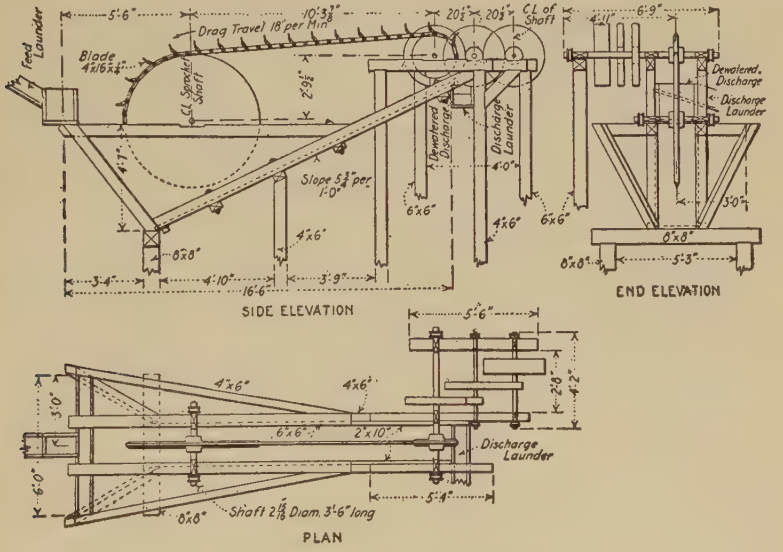

FIG. 5.—Federal-Esperanza classifier.

being held high enough to avoid crushing the cereal. Oats are commonly hulled in this way. Corn for hominy can be hulled by steeping in weak caustic potash followed by thorough washing. The coats are readily loosened by this treatment.

SEPARATION OF SOLIDS FROM LIQUIDS. LIQUIDS FROM LIQUID

The separation of coarse solids from water and other liquids can best be done with revolving screens if the least loss of head room is required. If loss of head room be not a consideration inclined stationary screens may be used. A convenient form of this kind is the catenary screen. To work out the curve of the supports for this screen, bend to a right angle a piece of screen cloth with mesh opening small enough to prevent passage of solids. Tack to either end of the bent screen a piece of thin board, forming a trough. Rest this device on a support with the angle of the screen down and the legs of the triangle projecting up into the air and firmly held by blocks placed under them. The solids with the operating proportion of water is discharged into the device

until it is filled up with grains, the water passing through them and the screen. The screen is then rotated about the vertex of the triangle until the dewatered solids slide freely over those in the pocket formed by the device, and the angle of slide is noted. This angle should be laid out on a piece of cardboard which will form a full sized template for sawing out wooden strips, to which the screen cloth will be secured for support. Secure the piece of cardboard to a wall and fasten at the upper end a piece of chain cutting the slope line. Draw the lower end of the chain up into a catenary loop securing it at some lower point of the slope line. The triangle formed by the original angle of repose plus the catenary bulge in its hypotenuse forms the template referred to. Cut out several pieces by it and tack the screen cloth over them. The first material fed into the device fills up the catenary pocket after which the solid material slides over the first solids fed in, which form the filter bed. Any desired depth of filtering pocket can be secured or a series of pockets may be laid out increasing in depth from top to bottom.

Devices for raking or scraping solids out of shallow tanks used for crystallization, precipitation or settlement, employ endless chains with flights mounted upon them or flight-pushing devices. In salt making these rise over the bed of salt in the pan on the back stroke and dip into it on the forward or advancing stroke. In removing salt from grainers the rakes in the latest and best reciprocating remover are actuated by a long stroke hydraulic cylinder. By the use of this means of actuation the number of linkage parts are reduced to a minimum, quite a consideration with the corrosive conditions which exist in salt houses. The hydraulic device is especially adapted to simply operating the rakes under the battery of grainer pipes in grainers. The discharge ends of the salt pans are inclined and the rakes drag the salt up the slope and out of the pan.

The separation of solids from liquids often involves at the same time the separation of solids from solids. In the so-called "classifiers" much used in Western ore mills the separation effected is watery sands from watery fines or slimes. The water content in the sands is of course very much less than in the slimes, the latter containing the bulk of the water fed to these devices. To reduce the water content in the slimes following their passage out of the classifiers they are often passed through thickening devices and the thick discharge from these is subjected to mechanical filter treatment and drying for further moisture reduction.

In the simplest forms of separation apparatus employing settlement with or without overflow of a product containing fines the settled material if sufficiently fine to pass a small opening can be drawn off continuously through a plug. This mode of disposition involves much escape of liquid and dilution of the discharging material and methods of avoiding this will be discussed at a later point in this section. If the material in the separating vessel is being elutriated as in preparing size from size stock means for discharging a charge of the material being extracted will be provided at the bottom of the separating vessel, the washing containing the substance being extracted being removed by solution and siphoning.

Mechanical Classifiers. –These devices consist of a tank into which the material fed flows at one end and what does not settle in the tank overflows at the lower end. Mechanical means are provided for removing the settled material. These devices have attained much use in ore mills of late years but others of similar

kind have been in use in other industries for a much longer time. A classifier of the Federal-Esperanza type to be described later has been used for a long time in the manufacture of white lead for separating the uncorroded portion of the buckles following the corroding and grinding operations.

The three principal classifiers employed in ore milling are the Federal-Esperanza, Dorr and Akins, illustrated by Figs. 5, 6 and 7. The Federal-Esperanza machine consists of one or two sprocket chains passing around sprocket wheels and to which metal flights are secured. The material fed enters at the lower end and the slime overflows at the same point. The sands are dragged by the flights above the water

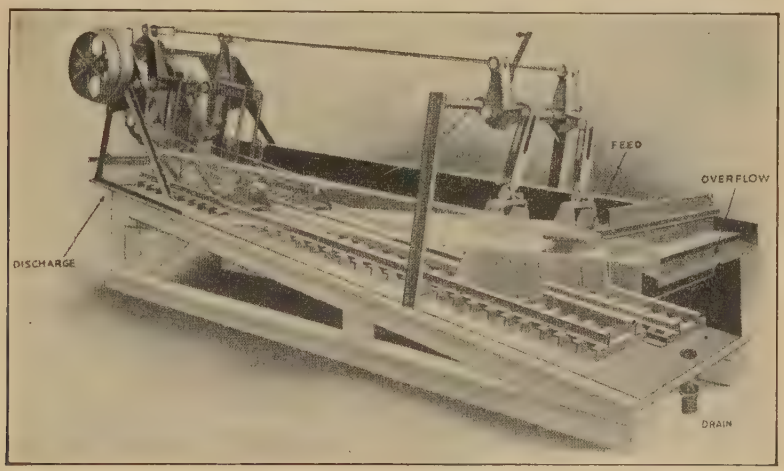

FIG. 6.—Dorr classifier.

in the settling tank in a launder extension, affording water and slime the opportunity to drain back and are discharged at the upper end. The Federal-Esperanza type may be designated as a homemade machine. It cannot be built more cheaply than the patented Dorr and Akins machine and on fine material the Federal-Esperanza does not do so good work as the patented machines but it can treat larger material. The writer has used them on rock material up to a size which would just pass a 10-in. opening. The use with the 10-in. material was for loosening clay coats preparatory to spray washing. The rock fragments had stood in dumps outdoors for nearly a half century and had accumulated clay coats which did not soften completely after being submerged 25 to 30 min. in cold water. In water of 95°C. the clay coats were thoroughly softened in 1 or 2 min. The period of 1 to 2 min. represented the time the material was in the classifier. The classifier tank was built of reinforced concrete. One very serious disadvantage of the Federal-Esperanza is the excessive wear of the chain pins and bushings if gritty material is being handled. The figure shows a light classifier with a single pair of main sprocket wheels. The principal designing point to be observed with these classifiers is to be certain that the larger main sprocket wheels at the bottom shall be of sufficient diameter so that the bearings can be placed above the water. The capacity per square inch of flight can be reckoned at 0.005 cu. ft. of dry material. The rate of travel of the chains and flights ranges from 15 to 30 ft. per minute.

The Dorr classifier uses a flight-pulling mechanism. On the upward or push motion of the blades they dip into the material which has settled in the enclosing tank and on the return motion they are raised above the bed of material being advanced. The back and forth motion of the rakes is given by a bell crank. The

raising of the rakes is done by cams which move the bell cranks on which the rakes are suspended through a certain arc at each revolution and hold the rakes above the settled bed on the return portion of the stroke.

The Akins machine as shown in Fig. 7 is a semi-circular tank on which is mounted a shaft supporting a spiral. The central portion of the spiral is all open except for

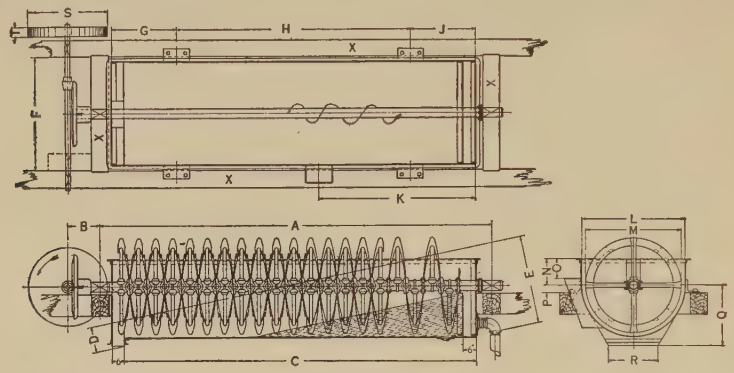

Fig. 7.—Akins classifier.

the spaces occupied by the arms which secure it to the shaft. The spiral is also broken at numerous points to permit of the draining back of slime and water in the upper portion of the classifier where the settled material has been wormed up above the ower or pool portion of the machine.

General Dimensions of Akins Classifier

Size, inches	*A*	*B*	*C*	*F*	*G*	*H*	*J*	*K*
16	75	8⅝	60	20	12	42	12	27
24	126	10⅝	108	32	24	66	24	46
30	144	10⅝	126	36	30	72	30	48
36	168	14⅞	150	40	30	96	30	66
45	182	12⅜	162	52	30	108	30	72
54	197	16⅜	174	61	42	96	42	81
60	221	16⅝	198	76	48	108	48	93

Size inches,	*L*	*M*	*N*	*O*	*P*	*Q*	*R*	*X*
16	18	16	3¾	5½	2	9½	....	2 by 4
24	27	24	8	12½	3½	15½	13½	6 by 8
30	33	30	9	14	3½	19½	16½	6 by 8
36	39	36	12	17	3½	22½	19½	8 by 10
45	48	45	12	20	3½	27	24¾	8 by 10
54	57	54	13	25	5¼	31½	30	8 by 12
60	63	60	11	25	5	33½	34	8 by 12

The above is intended to supply the data necessary for including these classifiers in drawings. Dimensions *D* and *E* will vary according to inclination of the classifier.

Engineering Data on the Classifiers.—Complete data on the performance of these machines is very difficult to obtain. The data given below is the most complete for the installations given that it has been possible to gather. Much of the published and private data which can be obtained is lacking in some important and essential feature and it is not deemed worth while submitting.

UTAH COPPER COMPANY

54-in. Standard Akins Classifier

Feed:

Tonnage to classifier per 24 hr., 1,050 tons dry.

Per cent of solids	37.50
Per cent of moisture	62.50
Ratio of water to solids	1⅔

Products of Classifier:

Sands raised 630 tons per 24 hr., dry.

Per cent of solids	60.63
Per cent of moisture	39.37
Ratio of water to solids	0.65

Slimes overflowing 420 tons per 24 hr., dry.

Per cent of solids	25.16
Per cent of moisture	74.84
Ratio of water to solids	2.98

Screens				Feed		Sands		Slimes	
Openings		Mesh	Diameter wire, inches	Per cent weight of material	Per cent weight accumulative	Per cent weight of material	Per cent weight accumulative	Per cent weight of material	Per cent weight accumulative
Inches	Millimeters								
0.263	6.680	3	0.070	0.08	0.08	0.55	0.55		
0.185	4.699	4	0.065	0.06	0.14	0.17	0.72		
0.131	3.327	6	0.036	0.36	0.50	0.70	1.42		
0.093	2.362	8	0.032	0.52	1.02	1.16	2.58		
0.065	1.651	10	0.035	1.19	2.21	2.63	5.21		
0.046	1.168	14	0.025	2.22	4.43	4.62	9.83	0.07	0.07
0.0328	0.838	20	0.0172	5.36	9.79	9.92	19.75	0.33	0.40
0.0232	0.589	28	0.0125	11.13	20.92	20.72	40.47	1.25	1.65
0.0164	0.417	35	0.0122	16.35	37.27	21.40	61.87	7.04	8.69
0.0116	0.295	48	0.0092	9.91	47.18	10.17	72.04	9.16	17.85
0.0082	0.208	65	0.0072	10.02	57.20	7.81	79.85	9.70	27.55
0.0058	0.147	100	0.0042	6.07	63.27	3.60	83.45	7.94	35.49
0.0041	0.104	150	0.0026	4.74	68.01	2.52	85.97	7.63	43.12
0.0029	0.094	200	0.0021	3.79	71.80	1.28	87.25	5.23	48.35
.....	....	280	.....	.....	.....	0.63	87.88	2.53	50.88
.....	....	through 280	.....	28.20	100.00	12.12	100.00	49.12	100.00

United Eastern Mill

Special 3 ft. 0 in. Wide by 14 ft. 8 in. Long Dorr Simplex Classifier

Feed:

Tonnage to classifier per 24 hr. 610 tons, dry. (This classifier working in closed circuit with a ball mill. Original rate of feeding 280 tons per 24 hr.)

Slope of classifier, 3 in. per foot.

Speed, 34 strokes per minute.

Per cent of moisture	39.2
Per cent of solids	60.8
Ratio of water to solids	0.65 : 1

Products of Classifier:

Sands raised 330 tons per 24 hr., dry.

Per cent of solids	80.0
Per cent of moisture	0.0
Ratio of water to solids	0.25 : 1

Slimes overflowing 280 tons per 24 hr., dry.

Per cent of solids	47.5
Per cent of moisture	52.5
Ratio of water to solids	1.11 : 1

Screens				Feed		Sands		Slimes	
Openings		Mesh	Diameter wire inches	Per cent weight of material	Per cent weight accumulative	Per cent weight of material	Per cent weight accumulative	Per cent weight of material	Per cent weight accumulative
Inches	Millimeters								
0.263	6.680	3	0.070						
0.185	4.699	4	0.065						
0.131	3.327	6	0.036						
0.093	2.362	8	0.032						
0.065	1.651	10	0.035	4.4	4.4	6.4	6.4		
0.046	1.168	14	0.025	12.2	16.6	18.3	24.7		
0.0328	0.833	20	0.0172	11.6	28.2	17.9	42.6		
0.0232	0.589	28	0.0125	9.6	37.8	15.8	58.4		
0.0164	0.417	35	0.0122	7.8	45.6	12.3	70.7		
0.0116	0.295	48	0.0092	5.2	50.8	10.8	81.5	1.2	1.2
0.0082	0.208	65	0.0072	5.0	55.8	6.2	87.7	3.0	4.2
0.0058	0.147	100	0.0042	4.8	60.6	4.5	92.2	6.4	10.6
0.0041	0.104	150	0.0026	3.8	64.4	2.1	94.3	6.6	17.2
0.0029	0.074	200	0.0021	0.4	64.8	0.6	94.9	5.2	22.4
.....	....	through 200	.....	35.2	100.0	5.1	100.0	77.6	100.0

The question of the capacity of the Dorr and Akins machines for any mesh separation desired in terms of dry tons is a question which cannot be answered owing to the number of varying factors affecting tonnage. Among these factors are the proportion of moisture entering the classifiers, the screen analysis of the material entering, the readiness with which the finer portions will settle, etc. The tabulation given below is approximately correct for the varying factors which it illustrates. To con-

TIMBER BUTTE MILLING CO.

45-in. Submerged Type Akins Classifier with Overflow Raised 10 in.

Feed:

Tonnage to classifier 350 tons per 24 hr., dry. (Feed consists of tailings from Wilfley roughing tables, Zinc department of mill.)

Slope of classifier 2 7/16 in. to 1 ft.; 8 r.p.m., power required to drive 1.39 hp.

Per cent of solids	41.40
Per cent of moisture	58.60
Ratio of water to solids	1.42

Products of Classifier:

Sands raised 217 tons per 24 hr., dry.

Per cent of solids	76.3
Per cent of moisture	23.7
Ratio of water to solids	0.31

Slimes overflowing 133 tons per 24 hr., dry.

Per cent of solids	23.7
Per cent of moisture	76.3
Ratio of water to solids	3.2

Screens				Feed		Sands		Slimes	
Openings		Mesh	Diameter wire, inches	Per cent weight of material	Per cent weight accumulative	Per cent weight of material	Per cent weight accumulative	Per cent weight of material	Per cent weight accumulative
Inches	Millimeters								
0.263	6.680	3	0.070						
0.185	4.699	4	0.065						
0.131	3.327	6	0.036						
0.093	2.362	8	0.032						
0.065	1.651	10	0.035						
0.046	1.168	14	0.025	27.0	27.0	55.0	55.0		
0.0328	0.838	20	0.0172	11.0	38.0	13.0	68.0	1.9	1.9
0.0232	0.589	28	0.0125	9.0	47.0	9.0	77.0	9.6	11.5
0.0164	0.417	35	0.0122	7.0	54.0	5.0	82.0	10.3	21.8
0.0116	0.295	48	0.0072	7.0	61.0	4.0	86.0	9.8	31.6
0.0082	0.208	65	0.0072	6.5	67.5	2.0	88.0	8.4	41.0
0.0058	0.147	100	0.0042	5.0	72.5	2.0	90.0	7.2	47.2
0.0041	0.104	150	0.0026	5.0	77.5	1.5	91.5	6.8	54.0
0.0029	0.074	200	0.0021	2.5	80.0	1.5	93.0	7.5	61.5
.....	....	through 200	.....	20.0	100.0	7.0	100.0	38.5	100.0

vert the tonnage into cubic feet capacity multiply by 22. The power required does not exceed 0.01 to 0.02 hp. per ton of dry material treated daily.

In addition to separatory uses the machines may be used in series for crystal or coarse granular washing and as a log washer in cleaning natural phosphates. For these uses the machines would be placed one below another and the wash water from

the uppermost machine after overflowing would pass to the machine next below and be used for wash water for this second machine, etc., thus employing the principle of Ostwald of the least use of wash water.

CAPACITY OF DORR AND AKINS CLASSIFIERS PER FOOT OF WIDTH[2]

Mesh[1] of separation desired	Overflow capacity, tons dry per 24 hr.	Raising capacity, tons dry per 24 hr.	Dorr, strokes per minute	Akins, revolutions of spiral per minute	Dorr slope in. per ft.	Akins slope	Water to pulp in feed
28	65 to 85	130	25 to 30	15 to 20	3¼ to 3¾	3"	2:1
48	60 to 70	125	25 to 30	15 to 20	3 to 3¼		3:1
65	40 to 50	100	20 to 25	12 to 15	2¾ to 3		3.5:1
100	30 to 40	90	16 to 20	10 to 12	2½ to 2¾		4–6:1
150	20 to 30	75	12	8	2 to 2¼		6–10:1
200	10 to 20	60	12	6	2 to 2¼	2"	10–15:1

Separation by Settlement and Decantation or Overflow.—The chief advance in these devices has been in mechanical means for aiding in the discharge of the settled material. Long V-shaped tanks have been provided with valves which open intermittently, the mechanism being a pulley the face of which is wanting at one or more portions of the circle. During the revolution of the faced portion of the circle of the pulley the spring valve is held by it against the opening of the tank. When the recessed portion of the pulley rotates to a point facing an opening the spring pressure is removed, the valve opens and the contents of the tank discharge. With these tanks the material fed passes in at one end and overflows at the other. The material which settles is graded from coarse at the entry end of the feed to the very finest which is capable of settling at the discharge end. The intermittent device works poorly on 12-mesh material and begins to be uncertain in action with 20-mesh material owing to chokage unless oversize plugs are used when the escape of water when the valves are opened is so great that very little effect is secured from discharging intermittently. Plug openings range in size from ⅛ in. to ¾ in.

Continuous Cone.—As shown in Fig. 8 the stream of material enters the feed spout *A* and, after passing through the truncated cone *B*, the water and solid particles which it is desired to remove flow upward and out into the overflow launder. The heavy particles settle in the cone and form the basin. When the settled material reaches the outlet of *B* it obstructs it so that the water rises in *B* lifting the float *C*, thereby lowering the ball *G* from the spigot by means of the lever *D*, the link *E* and the valve arm *F*. Once the cone is filled up a balance will be obtained and the settled solids will flow out in a continuous stream but the

[1] This should be the average of the first two meshes which report in the percentage by weight column which gives a greater average than 2 per cent, the separation mesh being then assigned to the screen of most openings of the pair involved in the averaging. In the case of the Timber Butte figures the separation is at 65 mesh, and of the United Eastern and Utah Copper mills at 28 and 35 mesh respectively.

[2] The widths of the Akins machine are the diameters of their spirals or dimensions *M* in the table of dimensions. Standard Dorr machines are made in widths of 2 ft. 3 in. 3 ft. 0 in. 4 ft. 6 in. 7 ft. 8 ft.

quantity discharged will be controlled by the quantity of settleable material entering the device. Some figures will illustrate the work of the device.

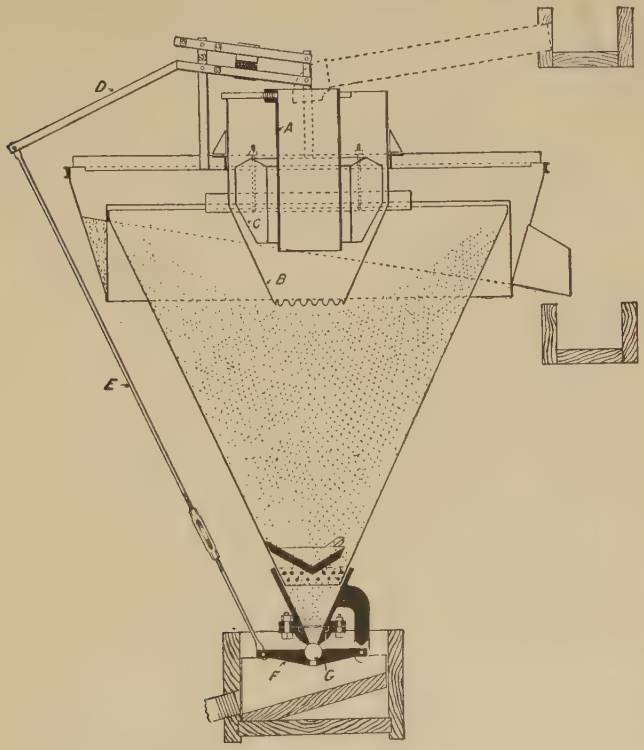

FIG. 8.—Continuous cone settler.

Example 1.—Feed to cone, 27 dry tons in 24 hr.

Moisture in feed, per cent	89.4
Moisture in plug product, per cent	28.6
Moisture in overflow, per cent	94.4

ABBREVIATED SCREENING TEST

Screen mesh	Per cent by weight feed	Per cent by weight plug product	Per cent by weight overflow
On 65	0.28	0.80	
100	4.66	9.39	0.31
150	18.00	31.90	1.33
200	14.75	23.97	3.76
Through 200	64.31	33.94	94.60

Example 2.—Feed to cone 200 dry tons in 24 hr.

Moisture in feed, per cent	75.4
Moisture in plug product, per cent	30.0
Moisture in overflow, per cent	89.6

Abbreviated Screening Test

Screen mesh	Per cent by weight feed	Per cent by weight plug product	Per cent by weight overflow
On 14	26.9	38.5	
20	10.7	15.2	
28	9.4	13.0	
35	7.2	9.8	
48	6.4	8.5	
65	3.9	5.0	0.25
100	4.0	4.4	2.75
150	3.3	2.4	5.00
200	0.4	1.0	3.50
Through 200	27.8	2.2	88.50

In the second use some of the material was too coarse to pass a 6-mesh screen. The large capacity is due to the coarseness of the feed. On material which will pass meshes ranging from 20 to 60 the rate of feed will range from 50 to 100 tons per 24 hr.

Thickeners.—These devices in point of fineness of material to be separated come below the classifiers. They are capable of giving clear overflows. With these devices liquids containing as low as 1 per cent solids can be thickened to a discharge containing as high as 40 per cent solids. With favorable factors the amount of solids in the discharge may be as high as 75 per cent.

The slope of the bottom of decantation tanks employing gravity discharge of the solids solely, ranges all the way from 0° to the 60° used in ore slime and paper pulp recovery work. Even with the extreme slope of 60° used in ore work there is an accumulation on the side of the tank by contact action which may reach a thickness of half an inch. Where the subsiding solids remain in the bottom of the tank as a suspense after slow settlement, a flat bottomed or nearly flat-bottomed tank may be used. Many organic compounds settle only to this degree particularly if they are in more or less of a colloidal state and if the percentage of solids is small it will be best to allow them to accumulate as long as possible so as to get the greatest compacting effect. If they are then in a condition to flow out freely a flat-bottomed or nearly flat-bottomed tank may be employed and the sludge can be discharged by a simple pipe and valve. If the solids do not flow out freely they can be removed periodically with hose and shovel, the tanks being provided with a system of drain cocks for removing the clear supernatant liquid.

The Booth water-softening apparatus for softening water by the soda-lime method employs a flat bottomed tank. The discharging means consists of one or two headers laying in the bottom of the tank and connected to a valve outside. Each header has a number of stub pipes screwed in at right angles on each side and open at their outer ends, the object being to uniformly reach, in discharging, all parts of the accumulated sludge.

Callow Tanks.—These tanks are illustrated in Figs. 9 and 10. For separation, where a clean liquid overflow is desired, the use of the smaller sizes shown in Fig. 9 is not to be recommended. As an inspection of the capacity table accompanying Fig. 9 will show the tanks have a small capacity and are suitable to small operations. Used in batteries they occupy more space than a single tank of equal settling area and capacity and there is the extra annoyance and expense in insuring the free running of the discharges. A battery of 50 8-ft. tanks requires not

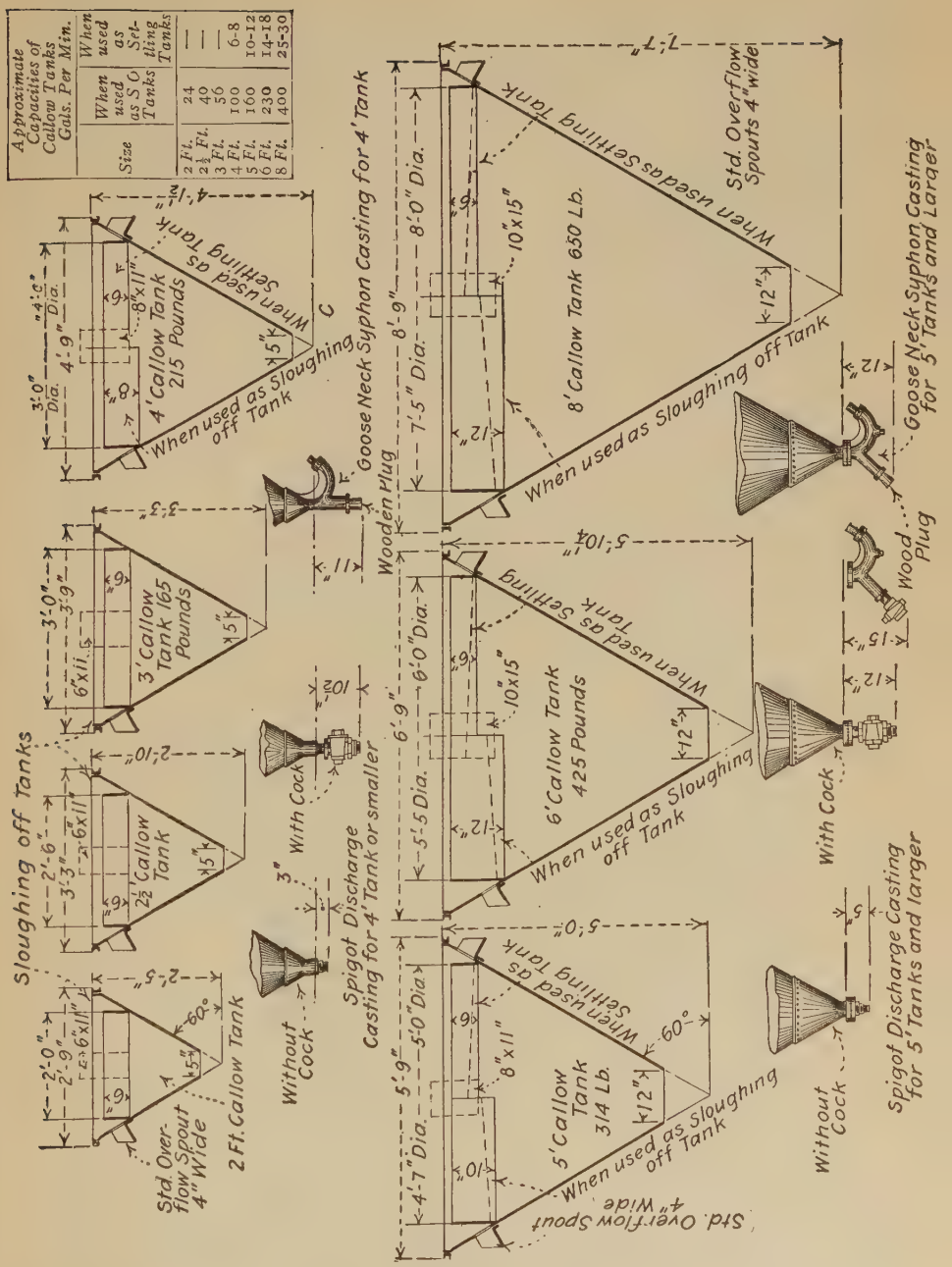
Approximate Capacities of Callow Tanks Gals. Per Min.
Size
When used as S.O. Tanks
When used as Settling Tanks
2 Ft. 24 —
2½ Ft. 40 —
3 Ft. 56 —
4 Ft. 100 6-8
5 Ft. 160 10-12
6 Ft. 230 14-18
8 Ft. 400 25-30
Sloughing off Tanks
2 Ft. Callow Tank
2½ Callow Tank
3' Callow Tank 165 Pounds
4' Callow Tank 215 Pounds
When used as Settling Tank
When used as Sloughing off Tank
5' Callow Tank 314 Lb.
6' Callow Tank 425 Pounds
8' Callow Tank 650 Lb.
Std. Overflow Spout 4" Wide
Std. Overflow Spouts 4" wide
Goose Neck Syphon Casting for 4' Tank
Goose Neck Syphon Casting for 5' Tanks and Larger
Wooden Plug
Wood Plug
With Cock
Without Cock
Spigot Discharge Casting for 4' Tank or smaller
Spigot Discharge Casting for 5' Tanks and larger

FIG. 9.—Callow tank thickening devices.

less than 4,375 sq. ft. of ground space while a single tank of equal settling area requires about 2,500 sq. ft. of floor space.

The goose neck discharge makes for reduction in head room but tends to choke, this tendency increasing as the material discharged becomes coarser. Most users of the tank prefer to run it without the goose neck extension and to throttle down by valves or plugs at the bottom of the tank with openings of suitable size. The loss of head room causes no reduction in density of pulp. Where the goose neck extension is used a proportionately larger discharge opening must be used than would be indicated by the less head merely owing to the necessity of providing for extra friction in the goose-neck extension and reasonable freedom from chokages likely to occur from passing the discharging material through the comparatively great length of small pipe.

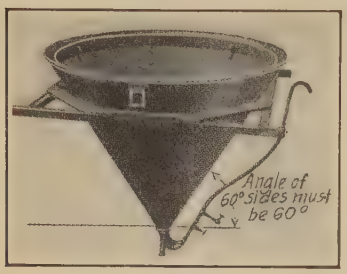

FIG. 10.—Callow tank.

For many years there existed much confusion of ideas among users of thickening apparatus as to the proper conditions of discharge. It was thought that a small discharge opening, the smaller the better, increased the thickening effect and also since the velocity of discharge increased as the head became greater it was thought

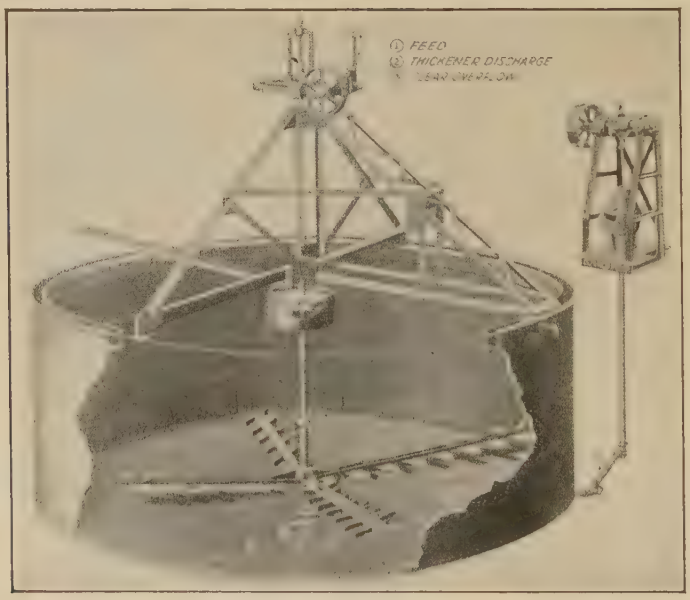

FIG. 11.—Dorr thickener.

that the head of discharge should be as small as possible. In any settlement apparatus there are zones of settlement, the rate in the zones near the surface of the liquid being fastest and that of the bottommost being slowest. In the lowest zone there is the greatest density of pulp and the slowest settlement and compacting of the settling material. It is evident then that if R is the rate of formation of this lowermost zone and A the area of the tank in which the material is being settled then RA is the cubical content rate at which the settling material can be discharged in its greatest

density. If V is the velocity of discharge and a the area of discharge opening then RA must equal Va. If V is fixed by the conditions as to head then evidently only one dimension of width of discharge opening will produce satisfactory operation of the thickening tank. If the head of liquid is reduced the discharge opening must be made correspondingly larger.

The loss of head room by not using the goose-neck extension only applies where one or a few of the cone tanks are used. Where a battery of tanks is used the head room required from entry point to lowermost distributing point of thickened material will probably be greater than in the device next to be described.

Dorr Thickener.—The tank and the distinctive Dorr mechanism assisting in remvoing the settled solids is shown in Figs. 11 and 12. The essential mechanism is plow blades set at an angle on arms secured to a slowly revolving shaft

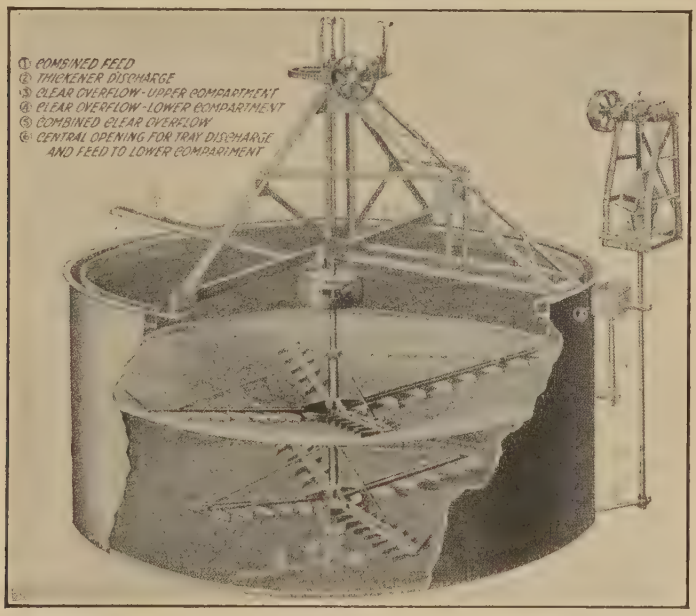
① COMBINED FEED
② THICKENER DISCHARGE
③ CLEAR OVERFLOW-UPPER COMPARTMENT
④ CLEAR OVERFLOW-LOWER COMPARTMENT
⑤ COMBINED CLEAR OVERFLOW
⑥ CENTRAL OPENING FOR TRAY DISCHARGE AND FEED TO LOWER COMPARTMENT

Fig. 12.—Dorr thickener with double raking device.

(from one revolution in 2 min. to one revolution per hour, the finer and more difficultly settleable the material the slower the rotation). As the mechanism rotates the settled material is pushed towards the discharge point at the center. It is claimed that the rotating mechanism has no harmful effect in stirring up the settling solids and that it assists in densifying the lowermost layers of settled material. Figure 11 shows a tank with single raking and by use of double raking mechanisms as shown in Fig. 12 the capacity of tanks of equal diameter is practically doubled. The tanks are made with the standard mechanism as shown up to 100 ft. and with the use of trays or double mechanisms up to 75 ft. Tanks with special mechanisms have been built up to diameters of 200 ft. If motor-driven, from 2 to 5 hp. should be provided for the standard sizes up to 100-ft. tank diameter. While running the power consumption is very much less than indicated by these motor ratings. Above 100-ft. tanks the power will depend on the layout of the raking mechanism and the makers will have to be consulted.

Theory of Settlement in Circular Tanks and Their Capacity.—By circular settling tanks are meant ones which are fed at the center and overflow around the whole periphery. Where there is absolute free settling, a condition which will be approximated if the proportion of solids is so small as to have no appreciable effect on the density of the stream of material entering a circular tank, then the path of a particle entering the center of the tank will be given by the equation

$$y = \frac{mx^2}{2RV}.$$

Further conditions for satisfying this equation are that the particles entering the tank will fall from the point of entry to the bottom with a finite definite rate. Second that there is uniform flow in the tank from the center to the periphery and parenthetically a uniform rate of flow of all liquid particles from top to bottom towards the periphery. Under this conception all particles in vertical lines at any given distance from the center will after any given flow interval still be in vertical lines and have arrived at the same distance from the tank center. Perfect flow of this kind is impossible to obtain. The entering stream can be forced to the bottom of the tank but tends to rise to the surface and flow in all directions towards the periphery. If it is endeavored to distribute the entering stream uniformly from top to bottom at its entry point some success can be had by employing suitably arranged deflectors but after leaving the deflectors the current will tend to rise towards the overflow. The position of the overflow governs the direction of outflow of the material fed to a tank. In a deep circular tank with overflow at the periphery at the top, the bottom areas particularly at the periphery are stagnant. The last assumption under the formula is that all the entering stream passes out into the body through the vertical center line, that it is concentrated at this line before outward flow and that the particles whose path is sought may flow outwardly along this vertical line. At whatever point they leave it will be the origin of the curve. Where comparison of the settling capacity of various shaped tanks is to be made it is convenient to assume that the particle begins its outward path at the surface of the water and that the origin is at this point or better that this is the origin of outward movement of all the particles entering the various shaped tanks.

In the formula m is the downward rate of subsidence which may be taken for water from the accompanying logarithmic curve for galena and quartz. Merely read off the number corresponding to the log in a logarithmic table and the velocity V and diameter of grain D will be given in millimeters per second and millimeters. Other substances of different specific gravity will have full velocities proportional to the specific gravity of the two given, nearly (*Cf.* "Velocity of Galena and Quartz Falling in Water," *Trans. A. I. M. E.*, XXXVIII). R is the radius of the tank and V the velocity at the overflow. To obtain V the cubical contents Q of the entering stream in cubic feet per second is divided by the depth of water in the tank times $2\pi R$. The depth of the water can be obtained by adding to the depth of the tank at the overflow point the head of water obtained from the expression $h = \sqrt[3]{\frac{9Q^2}{32\pi R^2 g}}$ which is merely the theoretical weir formula for discharge with the quantity Q transposed and the member b^2, the width of the slot squared, converted into $2\pi R^2$.

The expression for the path of a particle in a rectangular tank is very simple: y equals $\frac{Vx}{m}$. The head of water at the discharge end can be computed from the theoretical formula for discharge.

In a cylindrical tank the horizontal velocity along or parallel to the axis of X can be obtained from the proportion $V : v_x = x : R$ or $v_x = \frac{RV}{x}$ the velocity of a particle at any distance x from the center of the tank. Now $\frac{dx}{dt} = v_x = \frac{Rv}{x}$ and $xdx = Rvdt$.

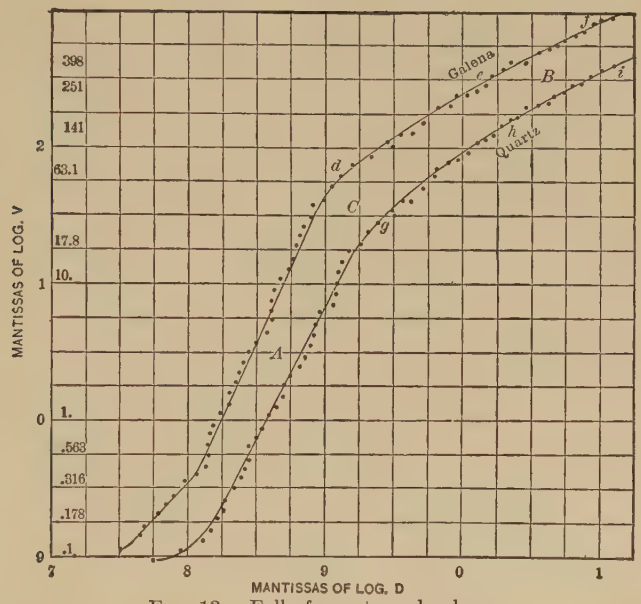

FIG. 13.—Fall of quartz and galena.

Integrating both sides of the equation $\frac{1}{2}x^2$ becomes equal to RVt or $t = \frac{x^2}{2RV}$, which gives the time for any particle to advance towards the overflow horizontally a distance x. In time t, however, the particle has subsided the distance mt or y. Substituting $\frac{y}{m}$ for t in the above time expression and transposing, the equation for path is derived as already given.

The various formulas given enable a comparison between circular and rectangular tanks on the point of capacity. Having determined the rate of subsidence of the particle which will reach the bottom at the periphery of a circular tank of any diameter by dividing the depth of water by time, $t = \frac{x^2}{2RV}$, the factor is obtained which will give the time for the same particle to traverse a rectangular tank of any cross section which has the same rate of feeding as the circular tank. It is only necessary to divide the factor into the proper depth of water of the rectangular tank to obtain this figure. The velocity of flow in the rectangular tank is obtained by dividing the rate of feeding by the cross sectional area of the water. This figure times the traversing time will give the length of rectangular tank equal in settling capacity to the circular one.

On these computations being performed for any hypothetical pair of tanks the areas of the two tanks are found to differ so slightly that for all practical purposes it can be stated that settling tanks of equal area have the same settling capacity. It is impossible to compute the exact head of discharge in each case because of the effect of the velocity of approach, the theory of correction for which is only an approximation and this question has not been considered in the discussion. In the case of the

rectangular tank the velocity of approach is the velocity in the body of the tank. The rectangular tank shows a greater depth of water at the discharge than the circular one because the overflow is more restricted and based on cubical contents; the equal settling capacity of the two shape of tanks is closer. One suspects that a complete theory would show that tanks of equal cubical contents would have equal settling capacity.

As a practical settling device the circular tank will be superior to the rectangular tank because of the quieter condition at the overflow, there being very much shallower overflow and consequently less tendency for converging currents to sweep partially settled material out of the tank. It is this quiet action at the periphery which gives its great advantage in settling material so finely divided as to be in the colloidal state or approaching the degree of subdivision that the phenomena of that state implies. If by the time the very finest particle reaches the periphery it is below the shallow depth of the overflow current then regardless of its rate of settlement it will eventually reach the bottom of the tank and can be discharged by means suitable for that purpose.

Capacity of Tanks on Very Finely Divided Matter.—The object of the separation between very fine matter and liquids is the production of a clear overflow and as great density as possible in the discharging thickened material. When relatively large proportions of finely divided matter and water are mixed and allowed to settle the mixture will be observed to arrange itself into zones or bands on account of a tendency of the particles to act rather as masses than individuals. The heaviest particles will settle promptly to the bottom of a glass test container while the balance will tend to arrange themselves in layers, each successive layer towards the top of the test vessel having ranges of grains with successively less average settling rate. As the zones settle clear water appears above the topmost one and the rate at which the surface of the topmost zone subsides is the practical rate of settlement for the material being tested. After a lengthy time it will be found that the subsidence of the upper surface of the top of the settling material practically ceases; when this condition is reached, the maximum density for thickened material has been reached and it is only necessary to withdraw some of the settled material and determine the proportion of dry solids to find out the greatest density of pulp which may be withdrawn from the settling tank.

In a settling tank with continuous entry of fresh material there is confusion of the zones owing to the fact that particles and flocs of material of comparatively rapid settling rate are settling through those of a slower rate of subsidence but as in the test vessel there is a lowermost zone whose boundary surface does not change so long as withdrawals are constant and there is no change in the character and quantity of the feeding stream. As must be evident there is a gradation in density measured from top to bottom of the settling tank ranging from that of the entering stream, or less, to that of the thickened material at the bottom.

In determining the capacity for Dorr thickeners the wet-crushed material is made up into a series of pulp mixtures with varying proportions of water and these are submitted to test for rate of settlement and the most practical maximum density of thickened material. The greater the proportion of water the faster the material will subside but on the other hand the greater the amount of clean water which must be removed, so that starting with a large proportion of water a large capacity will be shown owing to the greater influence of the higher settling rate and this will gradually diminish until a settling point is reached when the settling rate begins to show a stationary condition, that is, that the rate from one test to the next differs so little

that the diminished proportion of water to be removed begins to play an important part in the computations. When this point is reached the capacity begins to rise and the capacity figure at the turn represents the safe working capacity of the tank, one which will take care of all reasonable variations in proportions of water and solids entering a tank.

Let it be supposed that a test sample has been prepared showing a proportion of water to solids of 14.04 to 1 and that after 17 hr. the pulp has settled to 1.13 parts of water to 1 of solids and only to 1.12 to 1 after 24 hours, then evidently the economic point of thickening is 1.13 to 1. Settlement tests show an average rate of 1.78 in. per hour, hence there can be decanted 1.78 cu. ft. times 62.3 or 111 lb. of water per hour per square foot of tank surface. Since the feed contains 14.04 water to 1 of solids and the discharge 1.13 water to 1 of solids the overflow must contain the difference between the two ratios or 12.91 parts of water in the overflow to 1 part solids settled and is 111 lb. per hour. The solids consequently represent 8.59 lb. solids settled per hour or 206 lb. of solids in 24 hr. For a ton there would then be required 9.7 sq. ft. for 24 hr. which is the capacity of the tank with the dilution given.

The computations may be expressed in the following formula

$$\text{Square feet of tank surface required per ton dry per 24 hr.} = \frac{(F - D)\,2{,}000}{R \times 62.3 \times 24}$$

where R = Rate of settlement in feet per hour,
F = Parts of water to 1 part solids in feed,
D = Parts of water to 1 part solids in discharge.

The series of tests which follow the first one will give the following factors and capacities.

R	F	$F - D$	Square feet tank surface required per dry ton per 24 hr.
1.250	11.18	10.05	10.70
0.666	8.32	7.19	14.40
0.464	6.89	5.75	16.60
0.345	5.47	4.34	16.85
0.2331	4.03	2.90	16.65
0.150	2.607	1.477	13.10

The series of tests show that 16.85 sq. ft. of surface will be required for each ton settled.

With 8 parts of liquid to one of solids the settling area required by Dorr thickeners ranges from 5 to 25 sq. ft. per ton of solids.

Electrolytes.—By the use of electrolytes in the solution the settlement area may often be reduced and they can be used if they give results and have no harmful effect. Different acids and salts have varying flocculating effect. Alkalies in small percentages have a deflocculating effect. Less than 1.00 per cent of caustic potash has a decided deflocculating effect. Above this proportion the effect is flocculating. Very dilute salt solutions give a good flocculating effect but greater proportions do not cause any further gain in settling effect. Common salt solutions have not a very strong flocculating effect but the maximum effect is obtained when the concentration is about 0.10 per cent.

Uses of Dorr and Other Thickeners.—Outside of the separation use to which these devices can be put is their employment in what is termed "counter current washing." To show the extractive and recovery possibilities a hypothetical computation based on three tanks will be made. It is assumed for convenience that the tanks hold 110 tons of liquid and solid in the proportion by weight of 10 parts of liquid to one of solids. It is desired to save the liquid which is valuable and reject the solids with the minimum of valuable liquid. Let it be further assumed that the solids will settle to a ratio of liquid to solids of 1 : 1 in each tank.

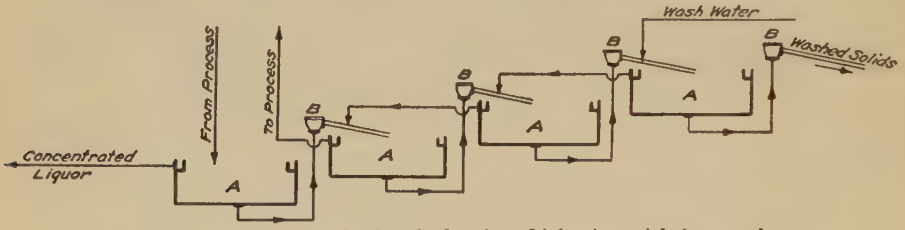

FIG. 14.—Diagrammatic sketch showing thickening with four tanks.

The mode of employing the three tanks would be to pump the underflow of the first tank to the second and dilute it with the overflow of the third. The underflow of the third tank goes to waste. Its feed is the underflow of the second tank plus the necessary wash water. The overflow of the first tank is finished liquid and goes to a storage tank from whence it may be drawn off for use or evaporation, crystallization or any operation to render it a usable or commercial product. The overflow of the second tank goes back into the main plant and after being used in making fresh solution will come back to the decantation apparatus again. The flows in the various tanks are indicated diagrammatically in Fig. 14 in which *four* tanks are engaged.

Since the ratio of liquid to solids is 1 : 1, 10 tons of 100 per cent liquid is removed with the solids from the first tank and passes to the second. Ninety tons of finished liquid pass at once from the first tank to the storage tank and there is consequently a 90 per cent saving effected in the first tank The 10 tons of 100 per cent liquid pass into the second tank along with the overflow of the third tank. This overflow consists of wash liquid of zero strength mixed with the liquid of very much less than 100 per cent strength which comes from the underflow of the second tank. If S'' is the strength of the liquid in the second tank expressed in percentage of that in the first tank and S''' that in the third then evidently

$$\frac{10 \times 100 + 90S'''}{100} = S'' \text{ and } \frac{10S'' + 90 \times 0}{100} = S'''$$

The first expression boils down to S''' equalling 1.099 when the value of S'' equals $10S'''$ obtained from the second equation is substituted. Consequently the strength of solution in the second tank is 10.99 and in the third 1.099. If two tanks only were employed the underflow of the second would carry to waste 1.099 tons or 1.099 per cent of the weight of the 100 per cent liquid run into the first tank or two tanks would effect an extraction of 98.90 per cent and three tanks effect a 99.89 per cent extraction. Having completed the computation it is only necessary to turn the tonnage figures used into rates of feeding to obtain the idea of continuous operation. This mode of decantantation arose in the cyanide process for extracting gold from ores. Of late years its use has been extended into other fields such as the continuous extraction of caustic soda from lime mud, in chlorine manufacture, as an auxiliary in the use of adsorbents such as silica gel, etc. (See also p. 354.)

Separation by Weighting.—This term should probably be limited to open tank operations where the agent of separation is a finely divided powder which on settling through the liquid in the tank carries down by simple attraction and entraining suspended solids. Perhaps as simple an example of this action as is to be found is described in U. S. Patent #251938 and others bearing on separating flocculent cotton as cellulose from acids in the production of celluloid and similar substances, the weighting down agent being powdered barium sulphate. This is introduced at the top of the open tank and in settling down through the acid carries with it the suspended particles of cotton. The other pole of separatory processes coming under this head would be chemical precipitation.

The action of some of the weighting agencies are purely adsorptive. Silica gel is used this way on some problems to which reference will be made later. Silica gel is a reversible adsorbent. It can be used as a filter as is bone char, one way of using fuller's earth, etc., or it can be used as a weighting agency in an open tank. Adsorption alone can produce important changes of state. In the preparation of "gray powder" mercury is triturated with chalk and becomes a fine gray powder from which is made mercurial ointment. The trituration is assisted by the holding apart action of the chalk. The small mercury particles finally resulting are prevented from coalescing by adsorbed gas transferred to them from the gas envelopes of the chalk particles.

Sewage.—With the great advance of the bacterial methods of disposing of sewage the agencies described below are fast becoming antiquated. In liming sewage (Etienne's patent, 1802) the lime first forms a carbonate and this in settling carries down suspended solids and organic matter which forms compounds of uncertain character. Magnesium chloride, aluminum sulphate and phosphate, barite and other salts have been proposed and used. In addition to the reagents clay, charcoal and other weighting and absorbent bodies have been proposed for use in connection with them. Aluminum salts form certain insoluble compounds with organic matter, the chemical and other actions being similar to those which take place in the manufacture of the lake colors. In the preparation of these colors to an infusion of organic coloring matter such as madder, cochineal, logwood or quercitron is added common alum which produces a precipitate of aluminum hydrate which in settling carries down the coloring matter forming the lake.

Adsorbents.—From the point of view of mechanical separation silica gel can be used in three principal ways. First as a filter (removal of moisture in vacuum-refrigerating plant, refining of paraffin, removal of vapors of alcohol, ether, benzol, acetone etc.) Second as a dust brought into contact with the vapor to be removed with recovery by cloth filtration (vapor from the blast of blast furnaces etc.) with activation by heating. Third by mixing the silica gel with a liquid to be refined allowing the silica gel with its impurities to settle, displacing the adsorbed impurities by water, filtering and activating by heat. This method of using the adsorbent is proposed for refining gasolene and kerosene and decolorizing operations. Counter current decantation fits in very nicely with this mode of employing the adsorbent.

That silica and silicates possess strong absorptive power has been known for a long time as witness the use of diatomaceous earth, fuller's earth, clay, talc, etc. for adsorptive, absorptive and decolorizing processes. Silica gel is prepared from

water glass by acids in a way that creates an enormous amount of ultramicrospic surface excellently affording in small compass an enormous surface power. As one authority has expressed it "If you consider the measure of the gel drop in centimeters, you must measure the area of the cracks in acres." This may give one clue to the adsorptive power of diatomaceous earth. Another clue to power of this kind in analgous substances may possibly be obtained from the following abstract of an article by Milton Whitney, Bureau of Soils, U. S. Department of Agriculture, appearing in *Science*, Oct. 14, 1921, p. 348, and taking up the subject of "Ultra Clay" isolated from ordinary clay. This ultra clay which appears to be a silicate of aluminum and iron will when dry absorb as much as 200 times its volume of ammonia gas, from 20 to 40 times of its weight of water vapor, and from 10 to 30 per cent of its weight of certain dyes. By heating the ultra clay to 900 to 1,000°C. this absorption power is practically completely killed. This ultra clay is as strong in its power of cementing sand grains as is Portland cement but when a dry briquette cemented by it is put into water it goes to pieces. The hardening of roads after a rain is attributed to it and their mushiness and slipperiness while wet.

Separation by Direct Electric Current.—This mode of separation forms the basis of numerous patents by Count Botho Schwerin (*cf.* "Control of Ore Slime," Oliver C. Ralston, *Eng. and Min. Jour.*, April 29, 1916). Many clay suspensions can be separated by utilizing the fact that particles of them suspended in water will migrate towards one or the other of the two electrodes introduced into the suspension. Direct current must be used and as clay particles will be negatively charged, they migrate towards the anode. In the purification of clays (*cf.* Ormondy, *Trans. Eng. Ceramic Soc.*, XII, pp. 36–64 (1912–13) and Bleininger, *Trans. Am. Ceramic Soc.*, XV, p. 335 (1913)) the impure clay is made into a slip of the consistence of thick cream and is preferably deflocculated by use of sodium hydrate. Coarse particles are settled out as well as some of the iron minerals. Following the settling operation the clay suspension is run into a metal trough into which dips a revolving cylinder made of lead or type metal. This is the anode. The current voltage ranges from 110 to 220 volts. The clay forms a blanket on the anode and after it revolves out of the liquid the clay is removed by a scraper. The clay contains only 17 per cent moisture which is less than would result with filter pressing under a pressure of 2 tons per square inch. The slip must be stirred during the separation operation in order to bring as many particles in contact with the anode as possible. According to the English authority cited the cost of current for depositing a ton of clay is in England from 10 to 64 cts. and a plant capable of producing 40 tons a week will cost $25,000. Successful use of this means of separation requires that the material to be separated be in an extraordinarily fine state of subdivision. Ralston reports that finely divided ores do not separate well by this method and attributes the failure to the fact that the ore particles are in too coarse a state of subdivision to be affected by the feeble motive force of the current. The successful separation of ground flint and kaolin and the failure with feldspar and ferric oxide described in the experiments below raises the query as to whether ores were tried which contained large proportions of the second pair of substances or other ones which do not separate well.

The movement of small particles under the influence of a direct current either to the cathode or the anode is called cataphoresis. Suspended particles of cotton, silk, starch, graphite, sulphur, etc., travel to the positive electrode. In pure water

most substances are charged negatively and hence move to the positive pole. Water itself is positive and moves toward the negative pole. No electro osmotic effects are noted in liquids like chloroform, ether, petroleum, oil of turpentine, etc. The addition of small quantities of electrolytes to solutions used in cataphoresis may promote endosmosis or oppose it. Large additions cause the action to cease entirely.

The addition of OH ions to a negatively charged particle should raise the cataphoretic migration velocity to the positive pole Acids and cations of higher valency would decrease it with increasing concentration and would finally change its sign (Freundlic, "Kapillarchemie," p. 239). Whitney and Blake (*Jour. Am. Chem. Soc.*, 26, p. 1339) found that upon adding 0.004 normal nitric-acid solution to colloidal silver the particles originally migrating to the positive pole reversed their direction.

There appears to be no relation between the amount of material separated and the current used and the separations are hence not governed by Faraday's law (*Tech. Paper* 51, U. S. Bureau of Standards, Washington, 1915).

In some experimental work (op. cit.) at the Bureau of Standards, Washington, to test out the Schwerin invention and to determine whether clays submitted to direct current separation would be improved by it, the conclusion was reached that most clays were not improved in quality by the separation and the success of the invention would lie in its being able to compete with the filter press. There seems some ground for believing that the process can remove such substances as pyrite from clays which tend to form dark spots or patches in the burned articles and it may have use in the clay and ceramic industries for removing substances not mentioned in the test work.

In the experimental work at the Bureau of Standards the positive electrodes were copper cans and the other poles pieces of carbon slightly dipping into the water of the test tank. The positive electrodes used were respectively 5 and 6 in. in diameter and were set in small tanks. Finely ground flint separated well at 125–180 volts and 0.75 amp. No electrolyte was necessary. No separation with or without electrolytes on Brandywine feldspar. Maine feldspar did a little better. Finely divided ferric oxide formed no deposit with or without electrolytes. An addition of 6 per cent of the oxide was then made to North Carolina kaolin. No reduction of iron content resulted. This illustrates one of the peculiarities of this mode of separation and which lies in the fact that if there is one predominating substance of a mixture which will move to a pole it will carry the inert substance with it.

Runs were made with North Carolina and Florida kaolin. Heavy firm deposits were secured with 0.2 NaOH and voltages ranging from 80 to 160. All matter in suspension was carried to the electrode. The densest layer is obtained with the highest voltage.

A fire clay of the No. 3 type from Aultman, Ohio, was ground to pass the 40 mesh size and made into slip showing minimum viscosity with 0.1 per cent NaOH. The separation took place very satisfactorily.

Upon making briquettes from the original clay, the suspended portion and the residue the linear drying shrinkages were found to be 8.4, 7.5 and 6.25 per cent respectively. Upon burning the specimens to cone 9 in a sagger the separated portion was decidedly superior in appearance, showing a uniform gray color while both the original clay and the residual part showed numerous black spots.

The reverse effect of the negative electrode was also investigated. Upon making a steel trowel the negative pole, it was found that water was drawn from a mass of plastic clay into which the tool was inserted causing the latter to penetrate very readily. All the products tested were submitted to preliminary screening and settling operations to remove coarse particles. Sodium hydrate and sodium oxalate were used as electrolytes.

A new process of water repelling-waterproofing of fabrics depends in part upon cataphoresis. The Tate electrical process of water-repelling waterproofing, as dis-

tinguished from the mode of waterproofing by rubber and gums which interfere with the ventilation of the fabric, and which is reputed to be an improvement on the chemical process of water repelling of treating the fabric with solution of sodium oleate followed by a bath of aluminum salt to form a metallic soap in the threads, consists essentially of an apparatus provided with graphite and aluminum electrodes between which and in close contact with which the fabric passes. The fabric before passing between the electrodes is saturated with a solution of sodium oleate while aluminum acetate flows down through grooves in the faces of the graphite electrodes. This saturates the fabric being treated and woolen pads which enclose the positive or aluminum pole. Electrolysis is set up and the resulting aluminum hydroxide is forced by cataphoresis through the woolen pad and through the fabric and tends to move towards the negative or graphite pole. In its passage through the fabric it reacts with the oleate and forms an insoluble basic aluminum oleate.

It is said that the chemical process does not firmly anchor the metal soap to the threads and that with use of the garment they become loosened from their anchorages. The ordinary aluminum oleate formed by the chemical process is soluble in the fluids used in dry cleaning, but the basic is not. Cataphoresis drives the aluminum hydrate particles into every part of the fabric. Tests on untreated cotton bags show leakage at the bottom in 1½ seconds, those treated by the chemical process analagous to the electric treatment in 31 seconds while those treated by cataphoresis showed no leakage from the bottom at the end of three weeks. On putting samples of cotton in dye liquor the untreated cotton picked up very little color, the chemically treated cotton became a deep red while the electrically waterproofed showed an even deeper shade. The electric treatment is said to increase the strength and luster of the fabric. ("Electrolytic Water Proofing of Textile Fabrics," by H. J. M. Creighton, *Franklin Inst.*, Oct., 1921.)

Osmosis and Dialysis.—The phenomena of osmosis and the method of employing them belong rather to the field of pure chemistry than chemical engineering but the phenomena of electro-osmosis seems suggestive and will be touched upon briefly. It was Porret who discovered that if a strong current is passed into liquids as though they were to be electrolyzed that there would be mechanical transport of the part of the liquid which becomes apparent if a porous diaphragm separates the electrodes by the liquid on one side of the diaphragm standing higher than the other. The phenomenon is most manifest with badly conducting liquids. The transfer of liquid particles is in the direction of the current and the liquid will stand highest on the cathode side of the diaphragm. Users of cells for the formation of sodium chloride and hydrate are familiar with electro-osmosis phenomena. If an anode side of a cell contains particles which would be negatively charged the liquid would migrate to the cathode and the solid particles to the anode.

Filters and Filtration.—In stationary or batch filters it is very seldom that the solids to be separated can be used as a medium and as is evident even in such cases there must be a support for the solids. The solids themselves do act as a medium to a certain extent even when prepared media are used. Prepared media range all the way through paper, cotton, and wool materials, biscuits of earthy materials, asbestos and slag wool, specially woven metal screens, to rock gravel and sand. Means of increasing the rapidity of filtration over the effect by gravity consist in applying suction or direct pressure by compressed air or steam or by mechanical appliances for exerting a direct pressure on the material being filtered. Surrounding the filter with hot water or a steam jacket will often

increase the rate of gravity filtration and is often particularly desirable where suction or pressure has the tendency to cause the solids to pass through the filtering medium.

Vacuum Filters, Continuous Type.—Vacuum leaf filters which dipped into a tank or basin of the material to be filtered were the predecessors in point of time of the continuous rotating vacuum filters of today, practically the only type of continuous vacuum filter which exists today. The old leaf type of vacuum filter was intermittent in operation and one of two sets of operations were necessary with them (omitting for brevity the washing operations). In one set the leaves were stationary and after the tank or basin was filled suction was applied and a cake of the proper thickness accumulated. It was then necessary to run off the balance of the material in the tank or basin, reverse the air current and blow off and wash out the cake which had accumulated. After this was done fresh material was allowed to enter and the cycle of operations was repeated. In the other set of operations and better from a mechanical point of view the filter leaves with their accumulated cake were removed by machinery and after the cake was dropped the leaves could be returned to the tank for further solid accumulation.

The open-tank suction filter is particularly adapted to cases where the amount of solids to be handled is only a small percentage of the total bulk; where the solids are very finely divided and where the cake takes a long while to build. There is but little wear on the filter cloth, and the cake builds to an even resistance, so that it washes evenly and with a minimum of washing.

According to the Industrial Filtration Corporation the rotary filter is indicated when the slurry will permit a cake ½ in. or more in thickness to build in 2 to 3 minutes when poured into a Buchner filter under 25 in. of suction.

The same corporation builds a hopper dewaterer for granular materials that will not form a cake. In this case a series of hoppers with filter bottoms pass successively under an overhead feed. The granular material is then dewatered by suction, may then be washed if desirable, and finally dumps underneath by gravity.

Figures 15 and 16 show the Portland filter which is similar in principle to other rotary filters and in general arrangement to the Oliver filter. The filter and its operation can be briefly described as follows: The material to be filtered is discharged into the wedge-shaped tank which supports the revolving drum. The air pipes at *A*, Fig. 15, blow air through the material to stir up the solids after a shut down. The connection to the valve plate at the center of the filter and above the pipe connections are not shown. Figure 16 shows a sprocket wheel near the drive pulley which drives a paddle agitator located in the bottom of the tank. In the interior of the drum will be seen the pipes which run from the panels forming the face of the drum and which are all connected in to the valve plate located at *B*, Fig. 15. The valve plate is shown diagrammatically in Fig. 17. *A* is the plate which is affixed to and revolves with the filter drum. It has as many holes in it as there are filter panels in the filter drum. *A* shows 22 holes which would correspond to the number of panels in a filter 12 ft. in diameter. *B* is the stationary port plate, fitting into a casting divided off into as many compartments as there are port groups. The openings at the front face of this casting are provided with pipe connections leading to vacuum and compression apparatus. For the majority of uses the most satisfactory machines for creating vacuum and compression are small compressors. To obtain vacuum the vacuum piping is connected to the intake end of one compressor and to obtain compression the compression-piping system is connected to the discharge of the other compressor. Compressors with easily removable or flat spring valves mounted in removable valve

plates make an excellent type for this kind of work as if anything goes wrong with the valves the removal of two or three stud bolts enables one to remove the valve

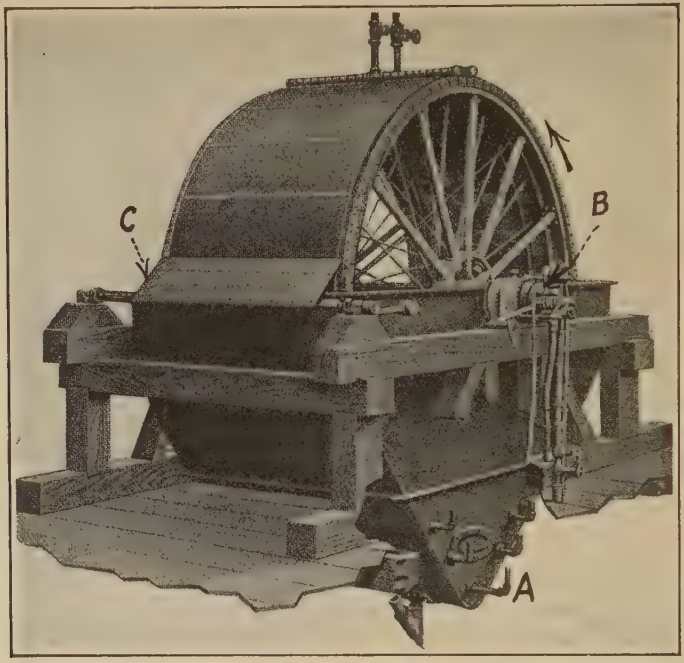

FIG. 15.—Portland filter, air side.

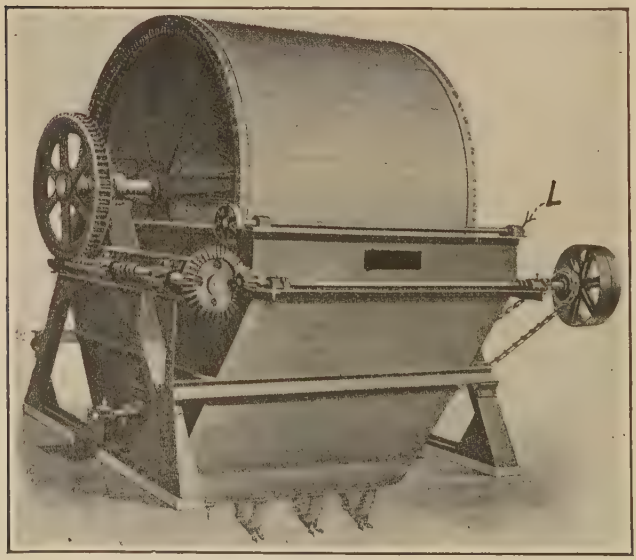

FIG. 16.—Portland filter, drive side.

plate and put in fresh springs the cost of which is trifling, the operation requiring no entry into the body of the machines.

The stationary port *B* has its inner face carefully ground so as to make intimate contact with the revolving plate *A*. The operation of the air passages is very simple. Port *C* connects all the panels which are submerged in the pulp and is under vacuum. *D* is a port into which all the pipes from the ascending panels open. *E* connects with all panels in the earlier part of their descent. *D* and *E* are under suction. *F* connects with each panel in succession just before it reaches the scraper *C*, Fig. 15, admitting air under pressure to dislodge the cake. *G* is another pressure port connecting with each panel after it passes the scraper to give the filter medium a cleansing action before becoming submerged. The valve plate of the Oliver filter is on similar lines.

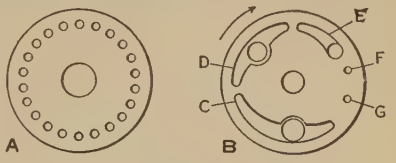

FIG. 17.—Portland valve plate.

If the cake is the valuable material to be separated ports *C*, *D* and *E* are all connected so that the worthless filtrate will go towards a common discharge point. If the filtrate is valuable *D* and *E* will carry wash water of two different strengths which may have different destination from one another and the strong liquor pulled through the port *C*. In the cyanide process for gold extraction weak cyanide wash may issue from *D* carrying with it further gold values from the cake while through *E* may pass the final wash water to remove any remaining cyanide.

Figure 18 illustrates the arrangement of the filtering medium on the Portland filter and shows the wooden frame construction. Where acid or caustic liquors have to be filtered the machines must be made of special materials and the filtering medium must be chosen from materials which will best withstand chemical action. The back of the panel is constructed of redwood staves held together with steel tongues. Each panel is made a complete unit by angle-iron segments bent to the curve of the drum and secured to the staves. The working face of each panel is provided with a molding *B*, making a recess in which the wire cloth, *C*, lies separated from the face of the drum. Within the space enclosed by the moulding, wooden strips, *D*, are fastened, these acting as supports for the wire cloth and providing drainage space for the liquid. The pipe which serves each panel has two connections with this space through the back, so that drainage is complete whether the panel is rising from the tank or descending into it. Grooves *E* lead to the pipe connections and assure unobstructed flow. Upon the wire cloth is placed burlap, indicated at *F*, and the surface fabric, *G*, surrounds this in turn.

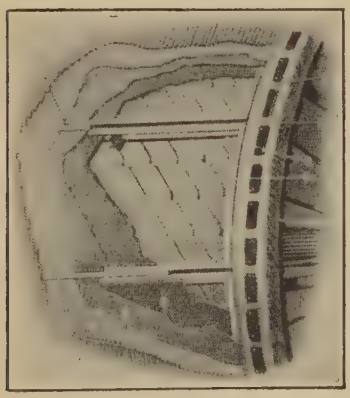
FIG. 18.—Detail of Portland filter.

The seal between the panels is formed in the space *H*. The canvas though continuous does not pass uninterruptedly from one panel to another but is carried down in the groove, *H*, and held there by a small strip which is squeezed into the canvas. This arrangement seals the panels from one another.

To hold the filter medium in place and protect it from wear by the scraper it is wound with wire. Figure 16 shows at *Z* a threaded shaft which assists in wire winding. The threaded shaft is belted up to the drum shaft. Wiring starts from the side of the drum and after the wire from the reel has passed through a tension clip or brake and around a groove of the threaded shaft it is secured to the side of the drum. As the latter revolves the threaded slot keeps it properly spaced and the brake insures

that it will be tightly wrapped on the canvas. A weight of wire of about 14 gage is suitable for winding the filtering medium. In the preparation of potash from Nebraska brines wire cloth has been substituted for the under cover of burlap.

Life of Filter Fabric.—For ordinary cold filtrations the filter fabric will last from 2 to 6 months or longer. With hot liquids which attack the fabric the cloth may only last this number of days. In cold Steffens house sugar work it may last from 21 to 50 days depending on the care of the cloth. In the cold work the saccharate of lime has a tendency to blind the pores of the filter cloth and it is necessary to scrub it with a weak solution of hydrochloric acid every 3 days and this necessary operation shortens the life of the filter cloth.

For most medium uses cotton fabrics give the best service. Wool and woven metal coth is also used. For ordinary cold work an 8 to 16-oz. duck or its equivalent in other cotton fabrics will show the best results. Other things being equal the more open the fabric the better the filtration results. If a little cloudiness in the filtrate is not objectionable it is better to equip the filter with an open fabric, preferably of medium weight if the greatest possible capacity is desired.

The Portland filter is built in drum sizes of 6, 8, 12 and 14 ft. diameter and with various width of face up to 16 ft. The Oliver filter is built in the following sizes of filter drum:

Diameter	Length	Area, square feet	Diameter	Length	Area, square feet
3 ft. 0 in. by	0 ft. 6 in.	4	8 ft. 0 in. by	10 ft. 0 in.	250
3 ft. 0 in. by	1 ft. 0 in.	9	8 ft. 0 in. by	12 ft. 0 in.	300
3 ft. 0 in. by	2 ft. 0 in.	18	11 ft. 6 in. by	8 ft. 0 in.	288
3 ft. 0 in. by	4 ft. 0 in.	36	11 ft. 6 in. by	10 ft. 0 in.	360
4 ft. 0 in. by	2 ft. 0 in.	25	11 ft. 6 in. by	12 ft. 0 in.	432
4 ft. 0 in. by	4 ft. 0 in.	50	11 ft. 6 in. by	14 ft. 0 in.	504
4 ft. 0 in. by	6 ft. 0 in.	75	11 ft. 6 in. by	16 ft. 0 in.	576
6 ft. 0 in. by	4 ft. 0 in.	70	11 ft. 6 in. by	18 ft. 0 in.	648
6 ft. 0 in. by	6 ft. 0 in.	105	11 ft. 6 in. by	20 ft. 0 in.	720
6 ft. 0 in. by	8 ft. 0 in.	140	14 ft. 0 in. by	14 ft. 0 in.	616
6 ft. 0 in. by	10 ft. 0 in.	175	14 ft. 0 in. by	16 ft. 0 in.	704
6 ft. 0 in. by	12 ft. 0 in.	210	14 ft. 0 in. by	18 ft. 0 in.	792
8 ft. 0 in. by	6 ft. 0 in.	150	14 ft. 0 in. by	20 ft. 0 in.	880
8 ft. 0 in. by	8 ft. 0 in.	200	14 ft. 0 in. by	24 ft. 0 in.	1,056

Capacity per Square Foot of Filtering Area.—Accoring to the Oliver Co. the dry solids of various kinds which can be obtained from a square foot of surface are: For sulphate of lime in phenol manufacture, 750 lb.; 200 lb. of molasses in cold saccharate work, 1,500 lb. of molasses in hot work, 3,500 of bicarbonate of soda, lime mud in caustic extraction, 750 lb. per 24 hours.

On cement slurry, ores and flotation concentrates the capacity will range from 200 to 1,300 lb. per 24 hr. per square foot of filter surface. Many plants will run under the smaller of these two figures through overestimation of the size of filter required, irregularities of operating, etc. Where the lower figure is reached with steady operation and giving the filter all the material it will handle and thickened as much as possible the limits of vacuum filtration for the substances mentioned have been

reached. If only 200 lb. capacity can be obtained then there is undoubtedly tremendous difficulties in settling and thickening the material for the filtration work and it has approached in state of fine subdivision the ceramic clays for the preparation of which the thickening devices which have already been described and vacuum filters are but little suited.

Percentage of Solid Passing to Vacuum Filters.—This should be where possible not less than in the proportion of 1:1. The thicker the material to be filtered flows to the filter the greater its capacity. This is shown by the results obtained by grinding a copper ore to pass an 80-mesh screen. This yielded a granular material which settled and filtered readily. The tests were made on a filter leaf 11 by 13 in. and the cake was allowed in each test to load up for a period of 3 minutes and vacuum was maintained for an additional 2 min. to dry the cakes formed.

Test number	Liquid, per cent	Vacuum, inches	Cake			
			Wet weight, ounces	Dry weight, ounces	Thickness, inches	Moisture, per cent
1	75	15	6	4½	⅛	25.0
2	50	16	44	33½	⅝	23.8
3	39	17	78	59	¾	24.3

The exigencies of a process may however require a large proportion of liquid. Thus in cold saccharate work the molasses is diluted with water before the introduction of lime and the material filtered only has about 10 per cent solids in it. The calcium saccharate cake made has about 65 per cent moisture in it, which is not objectionable since carbonating the cake to release the sugar follows the filtering operation.

Thickness of Cake and Moisture Content.—Where there are irregular conditions the thickness of the cake may be as little as ⅛ in. The ordinary cake thickness ranges from ¼ to ¾ in. A cake of greater thickness can be made if it is built up of extra porous material. The more finely divided the solids are and the more colloidal they are the thinner the cake must and will be and unfortunately the greater will be its moisture content.

The moisture content is commonly understated. Crystalline solids will often make a cake under 10 per cent moisture but amorphous solids will make a cake running from 10 to 20 per cent moisture and up to as high as 30 per cent or more. On colloidal material the moisture may run up to 30 per cent and not show free moisture after being scraped from the filter and piled. The range of moisture in Western flotation concentrates is from 10 to 20 per cent, occasionally higher. With this substance yielding a high percentage of water in the cake the aim is to reduce the moisture only to the point where water will not separate out in the railroad cars. Drying of material of this kind is so difficult and costly that it is not frequently attempted.[1]

[1] In a recent appearing device for drying this sort of material the slime is forced through a large plate pierced with humerous holes resembling a similar plate used in meat grinders. The forcing of the slime through the holes is secured by the same means as is employed in meat grinders, a worm. The slime is formed through the pierced plate like a great mass of spaghetti and issued at the same rate of travel as the receiving steel plate conveyor placed below the pierced plate. The conveyor passes through a drying oven. The object of course of this invention is to promote drying by creating a large amount of drying surface.

The two systems of vacuum filtering that may be practiced, are termed "wet" and "dry." In the wet system the liquid drawn through the filtering medium passes through the vacuum apparatus. In the dry means are employed between the filter and vacuum apparatus for removing the liquid. The wet system does not permit of maintaining a high vacuum since there must be sufficient clearance of the moving parts of the vacuum apparatus to handle liquid. If there be grit in the filtrate the moving parts of the vacuum apparatus in contact with the filtrate will be attacked. Rotary vacuum pumps are commonly employed in the wet system and it has the advantage of simplicity and can be employed to advantage on solids which filter readily and with clear and non-corrosive filtrates. A vacuum pump made of acid-resisting materials adds further cost and complication.

The various pieces of accessory apparatus can of course be placed where they are most convenient and where a battery of filters is connected into the accessory apparatus the only point to be observed beyond securing the necessary capacities throughout the accessory apparatus is to see that the piping has sufficient fall to secure drainage. The vacuum receivers in the dry system are fitted with a float and vacuum release so that if anything goes wrong with the solution pumps the vacuum apparatus will not be flooded.

It is usual to allow ¾ cu. ft. of displacement in the vacuum apparatus to every square foot of filtering area. To obtain the displacement of a compressor subtract one-half the net cross-section of the piston rod from the area of the cylinder, convert the latter into square feet and multiply by twice the stroke expressed in feet or fractions of a foot and the number of revolutions per minute. The "free air" capacity of compressors at sea level as given by the manufacturers is the displacement and can be used as a practical factor in determining the proper size compressor to use for vacuum purposes.

From 15 to 25 in. vacuum is maintained depending upon altitude. The higher the altitude the less the vacuum which can be maintained. The appended table shows the relation between atmospheric pressure in pounds and mercury reading in inches. Inches divided by 2.04 will give pounds.

Atmospheric Pressure and Barometer Readings at Different Altitudes, Approximate Values

Altitude above sea level, feet	Atmospheric pressure, pounds per square inch	Barometer reading, inches of mercury	Altitude above sea level, feet	Atmospheric pressure, pounds per square inch	Barometer reading, inches of mercury
0	14.7	30.0	5,500	11.9	24.3
500	14.4	29.4	6,000	11.7	23.8
1,000	14.2	28.8	6,500	11.5	23.4
1,500	13.9	28.3	7,000	11.2	22.9
2,000	13.6	27.7	7,500	11.0	22.5
2,500	13.4	27.2	8,000	10.8	22.1
3,000	13.1	26.7	8,500	10.6	21.7
3,500	12.9	26.2	9,000	10.4	21.2
4,000	12.6	25.7	9,500	10.2	20.8
4,500	12.4	25.2	10,000	10.0	20.4
5,000	12.1	24.7			

Power Required.—In compression work the pressure on one side of the piston is the atmospheric or very close to the atmospheric and on the other the outlet

pressure which is being maintained at so many pounds above atmospheric. In vacuum work we have the residual pressure due to maintaining the vacuum and on the other side the atmospheric. In other words the compressor is working against a pressure equal to gage pressure converted into pounds. At low pressures there is very little difference in the horsepower required to compress air at sea level or high altitudes. For the majority of vacuum problems 6 to 7 hp. per 100 ft. of vacuum displacement will cover the filter, the vacuum compressor, the pressure compressor and the centrifugal pump or about 5 hp. for each 100 ft. of filtering surface. The pressure compressor requires much less power than the vacuum machine. In many filtration problems it is only required for the occasional blow necessary to free the filter medium of particles which have clogged it and the small use of air below the scraper to the same end, for blowing the cloth after a shutdown or for stirring up the material in the filter tank after a shutdown. Its use to blow the cake before scraping increases the moisture as it tends to blow through a part of the moisture adhering to the burlap and screen under the filtering medium and its use at this point should be avoided. The power to drive the filter is small, less than 1 hp. on machines under 12 by 12 size. Filter drums revolve from $\frac{1}{5}$ to $\frac{1}{10}$ r.p.m.

The American filter consists of a series of disc leaves placed close together and mounted on a revolving shaft.

The Robacher filter uses "filtros," an artificial porous stone as the filtering medium and mounts the sections on the sides of a rotating drum.

Costs.—Ten cents per ton would represent about an average cost for vacuum filtering including preliminary thickening. The cost of disposing of the product made will average about 20 cts. per dry ton. This figure is the cost of disposal by hand labor or loading for shipment. Where the cake is of no value or is subjected to further treatment mechanical disposal will make the cost figure very much lower.

Pressure or Intermittent Filtration.—The disadvantage of this mode of filtering lies in the intermittency of operations and the tendency to mussiness about the filter with consequent loss of filtrate and cake. The loss of capacity is offset to some extent by the ability to form a heavier cake. The principal advantage of pressure filters is in washing. Given any fixed and necessary amount of washing the pressure filter system can be arranged to suit. In the vacuum systems washing can only be performed between the time the drum rises out of the tank and until it reaches the scraper. It makes no difference whether the drum is run slowly or fast. If the drum is run slowly the cake is heavier and the rate of percolation through it of the wash is slower. It is useless to turn any more wash liquid on the vacuum cake than will be taken care of by the rate of percolation through. Volatile wash liquids cannot be used on vacuum filters without serious loss by evaporation or loss of one of its principal advantages, the ability of seeing the operation of filtering. On solid materials which hold moisture with great pertinacity or through which liquids percolate very slowly the vacuum filter does not work successfully and pressure filtering is required. Ries[1] cites a natural clay from Wyoming which when placed in a measuring flask basorbed and retained sufficient water to increase it bulk eightfold. Some

[1] Clays, Their Occurrences, Properties and Uses," John Wiley & Sons, New York, 1906, p. 128.

natural clays or the slips prepared from them may have a range of diameter of particle not much outside of 0.005 mm. and from that down to 0.001.[1] According to Ries[2] the classification of detrital material according to size would be beginning with a fineness represented by "gravel:"

Conventional name	Diameter	
	Inches	Millimeters
Gravel	1/12 to 1/25	2.0 to 1.0
Coarse sand	1/25 to 1/50	1.0 to 0.5
Medium sand	1/50 to 1/100	0.5 to 0.25
Fine sand	1/100 to 1/250	0.25 to 0.1
Very fine sand	1/250 to 1/500	0.1 to 0.05
Silt	1/500 to 1/2,500	0.05 to 0.01
Fine silt	1/2,500 to 1/5,000	0.10 to 0.005
Clay	1/5,000 to 1/25,000	0.005 to 0.001

The graphite and clay mixture used in lead-pencil manufacturing requires pressures up to 200 lb. and a pottery clay has already been cited which required 2 tons per square inch to reduce the moisture to 17 per cent. When one compares pressures such as these with the greatest possible pressure which can be obtained in vacuum filtration or less than 14.7 lb. (the atmospheric pressure at sea level) the advantage of pressure filtering in certain filtering problems is at once apparent. In the vacuum filter low pressure is offset to some extent by making a thin cake.

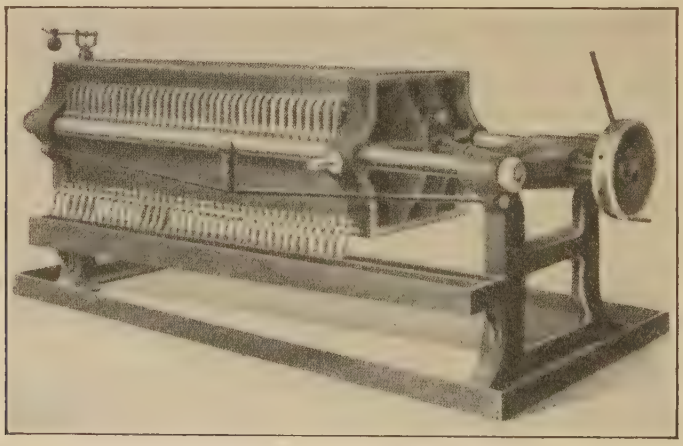

Fig. 19.—Square-leaf filter press.

In plate and plate and frame presses certain filtering media can be used which cannot be employed on vacuum filters. In clarifying certain vegetable oils, filter paper must be used and quite apparently such a medium cannot be used on a rotary vacuum filter.

[1] G. F. Merill, "The Non-metallic Minerals," p. 233.
[2] *Op. cit.*, p. 116.

Plate and Plate and Frame Presses.—All the usual procedures in filtering can be carried out with these devices. The leaves can be filled with material to be filtered until filtrate ceases to pass through the medium which is ordinarily canvas when the press must be discharged of its cake. Either the filtrate or cake is rejected. After the press is filled or partially filled various washes may be introduced and separately distrained if desired, the openings in the leaves being arranged to that end.

Figure 19 shows a square leafed press, the usual shape of leaf, of 30 by 30-in. section outside and with center screw for binding and releasing the leaves for loading and unloading.

The two great divisions of filter presses are into plate presses and plate and frame presses. The plate leaf is the supporting leaf for the filtering medium and through the openings of which the filtrate issues. The frame is the leaf into which the material to be filtered discharges. If plates only are employed they must be of the recessed type, see Fig. 20. In the recessed type of plate the depth of recess is about $\frac{5}{8}$ in. Referring to Fig. 20 the hole in the upper left-hand corner is the feed hole. To employ the recessed type a piece of cloth is cut long enough to cover both sides. A hole is cut in the cloth where it passes over the feed hole on each side. A gommet nut is then inserted in the hole and the screw rings are tightened up on it to make a seal.

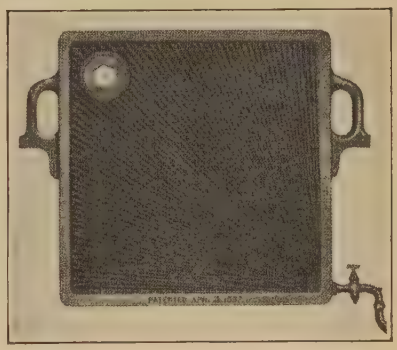

FIG. 20.—Recessed plate.

When all the plates are tightened up in the press the cross-section of the plate and the filtering medium will be as represented in Fig. 21. The material to be filtered passes into the wedge-shaped spaces formed by the filter medium. The filtrate passes through the cloth and out through the cocks as indicated by the arrow points. In order to wash with recessed plates, cake must not be allowed to accumulate until it fills the wedge-shaped spaces completely, but to a degree less than this so that the wash liquid will have a chance to pass through. The recessed-plate type of filter press should be employed where possible owing to its lower cost. It cannot be used with delicate filtering media such as paper because of the strain on such media at the points where it is bent. On the other hand if a very thin cake is necessary owing to the slow rate of percolation through some solids the recessed type may be used to advantage and work with relatively low pressures. With cakes from $\frac{3}{8}$ to $\frac{1}{2}$ in. the recessed type is indicated as it is impractical to make a frame so thin. The recessed type would not work advantageously with liquids which attack the filtering medium. The latter would give away under chemical attack at the points bent. As the medium lies straight in the plate and frame type of press it is better suited to coping with fluids which attack the filtering medium.

FIG. 21.

In plate-and-frame presses the leaves are assembled with the plates and frames alternately placed. Figure 22 shows the feed "eye" and part of the frame construction. Figure 23 shows the arrangement of plates, frames and filtering media in section. There is a section of cloth between each plate and frame. "Simple washing" such as has already been described can be employed with plate-and-frame

filters, if suitable. If deep frames are used and "through washing" is employed, that is introducing the wash fluid between the partition center of a plate and the filtering medium and forcing it through the cake to the similar recess on the other

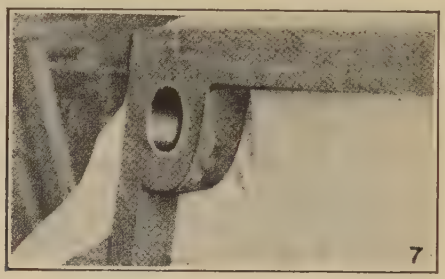

Fig. 22.—Feed eye of plate.

side, using every other discharge orifice or cock; then if the cake formed is of a loose character and offers little resistance to pressure or tends to build up slowly leaving open spaces in the center of the frame there will be a tendency to bulge the filtering medium and an excessive strain will be put upon it shortening its life. Simple washing is better in such cases. The section, Fig. 24, shows one way of attaining "through washing" by the "every other cock" method.

Separate filtrate and wash orifices or cocks should be employed where possible, as with this way of working if anything happens to the filtering operation at any

Fig. 23.

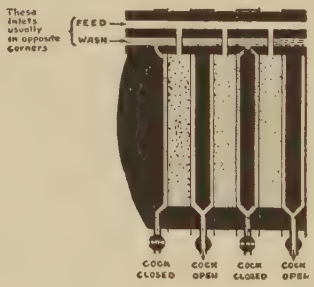

Fig. 24.—"Through" washing.

point it will be detected at once. If any leaf is filtering badly from irregularity of cake accumulation which can be detected by cloudiness in the issuing filtrate the particular cock or cocks involved can be shut off until the cake has accumulated to a thickness to yield a clear filtrate. With volatile liquids cocks cannot be used as the filtrate or wash of this kind must be led to a closed vessel to avoid loss by evaporation. If there is a volatile wash following the cake formation it will be desirable to have a special outlet for it. In case there are no cocks there are two channels, each of which drops in to alternate plates. In fact when special eyes for steam, etc., are also considered the possible combinations are almost endless and it is best to consult the manufacturers where there is an involved washing necessary, in order to obtain the most suitable leaves.

12-IN. PRESS

Number of plates	Filtering area, square feet	Volumetric capacity, cubic feet					
		Thickness of frames, inches					
		1¼ or recessed	1½	1¾	2	2½	3
15	24.5	1.28	1.53	1.79	2.05	2.56	3.06
20	32.2	1.68	2.02	2.36	2.69	3.31	4.02
25	39.9	2.08	2.50	2.91	3.33	4.16	4.98
30	47.6	2.48	2.98	3.48	3.97	4.96	5.90
35	55.2	2.88	3.45	4.02	4.60	5.76	6.90
40	62.8	3.27	3.93	4.58	5.24	6.55	7.85
45	70.5	3.68	4.40	5.14	5.88	7.36	8.81
50	78.1	4.01	4.89	5.68	6.52	8.14	9.76

18-IN. PRESS

Number of plates	Filtering area, square feet	Volumetric capacity, cubic feet					
		Thickness of frames, inches					
		1¼ or recessed	1½	1¾	2	2½	3
15	55.0	2.87	3.44	4.01	4.58	5.73	6.88
20	72.5	3.78	4.54	5.30	6.05	7.56	9.06
25	89.6	4.68	5.62	6.55	7.50	9.36	11.22
30	107.0	5.69	6.80	7.81	8.93	11.16	13.40
35	124.0	6.49	7.78	9.07	10.36	12.95	15.51
40	141.0	7.35	8.85	10.30	11.80	14.70	17.70
45	158.5	8.30	9.95	11.60	13.25	16.60	19.90
50	176.0	9.21	11.00	12.90	14.70	18.40	22.10

24-IN. PRESS

Number of plates	Filtering area, square feet	Volumetric capacity, cubic feet					
		Thickness of frames, inches					
		1¼ or recessed	1½	1¾	2	2½	3
15	103.0	5.35	6.44	7.50	8.57	10.70	12.84
20	135.0	7.04	8.45	9.85	11.25	14.06	16.89
25	168.0	8.70	10.45	12.20	13.94	17.40	20.88
30	199.0	10.39	12.46	14.52	16.60	20.75	24.90
35	231.5	12.05	14.49	16.88	19.30	24.10	28.95
40	264.0	13.75	16.50	19.25	22.00	27.50	33.00
45	296.0	15.40	18.50	21.60	24.70	30.80	37.00
50	328.0	17.10	20.50	23.90	27.40	34.20	41.00

30-in. Press

Number of plates	Filtering area, square feet	Volumetric capacity, cubic feet					
		Thickness of frames, inches					
		1¼ or recessed	1½	1¾	2	2½	3
15	168.1	8.76	10.5	12.25	14.0	17.55	21.06
20	221.0	11.52	13.8	16.10	18.4	23.00	27.63
25	273.0	14.25	17.1	19.95	22.8	28.50	34.20
30	326.0	17.00	20.4	23.80	27.2	34.00	40.80
35	379.0	19.75	23.6	27.60	31.6	39.50	47.40
40	431.0	22.42	26.9	31.40	36.0	45.00	53.85
45	484.0	25.20	30.2	35.20	40.5	50.50	60.40
50	536.0	27.90	33.4	39.00	44.8	56.00	67.00

36-in. Press

Number of plates	Filtering area, square feet	Volumetric capacity, cubic feet					
		Thickness of frames, inches					
		1¼ or recessed	1½	1¾	2	2½	3
15	251	13.1	15.7	18.3	20.9	26.2	31.40
20	330	17.2	20.6	24.0	27.5	34.4	41.25
25	408	21.2	25.5	29.8	34.0	42.5	51.00
30	486	25.3	30.4	35.4	40.5	50.6	60.70
35	565	29.4	35.3	41.2	47.1	59.0	70.60
40	643	33.5	40.2	46.9	53.6	67.0	80.40
45	721	37.6	45.1	52.1	60.1	75.1	90.10
50	800	41.6	50.0	58.0	66.6	83.4	100.00

42-in. Press

Number of plates	Filtering area, square feet	Volumetric capacity, cubic feet					
		Thickness of frames, inches					
		1¼ or recessed	1½	1¾	2	2½	3
15	347	18.05	21.7	25.3	28.9	36.1	43.3
20	456	23.70	28.5	33.3	38.0	47.5	57.0
25	564	29.40	35.2	41.1	47.0	58.7	70.5
30	682	35.50	42.6	49.7	56.8	71.0	85.2
35	781	40.70	48.8	57.0	65.1	81.5	98.0
40	890	46.40	55.6	65.0	74.2	92.7	111.0
45	1,000	52.10	62.5	73.0	83.4	104.0	125.0
50	1,107	57.60	69.2	80.7	92.2	115.0	138.2

The filling of filter presses and the maintenance of the proper pressure is done almost universally by pumps and montejus. Rotary pumps are most favored. Capacities for the square-leaved presses of one reputable maker are shown in the accompanying tables.

The Sweetland filter is a frame press arranged so that the bottom can be lowered as a unit in order to remove the cake quickly by washing it out.

The Vallez rotary filter consists of a number of circular leaves rotating about 1½ r.p.m. inside a tank which is kept under pressure. The effluent escapes through the shaft in which the leaves rotate. Washing, blowing and steaming are all possible with it. To dump the cakes the entire bottom of the body is unfastened. The filter appears to be popular in the beet-sugar industry.

Brewery Presses.—In the filtration of beer and other malt beverages following defecation, and before bottling or barreling, plate presses are used, the plates being arranged vertically or horizontally. Filter mass coming in cakes or sheets and consisting of compressed cotton or cellulose is the filter medium. In using the filter mass it is broken into pieces and disintegrated in water. After this is done it is poured into the recesses of the plates, a screen in the latter holding the medium in place. Before filtering the filter mass is compressed with a powerful screw. In the vertical presses the screw is a part of the press and the leaves are all compressed at the same time after the hinged cover of the press holding the screw is put on and bolted to place. (See Fig. 26.)

Screws for compressing the medium separate from the press are also used and on the point of securing uniformity in the layers of medium they seem preferable as each leaf can be handled separately and watched to see that a layer of medium of uniform density is obtained. This requires some skill and attention, the danger being that the medium will not be sufficiently compacted in some places leaving channels through which the liquid will pass. At least two presses are required and the moment the filtrate shows any cloudiness the malt liquor is switched from the press in use and pumped through a fresh one. When this occurs the press is unbolted and the filter mass is removed to a mechanical washer where with the aid of hot water is freed of yeast and other substances. After the washing it can be used over again. Some brewers now incline toward the use of continuous rotary vacuum filters.

Filter Aids.—Among these are calcium sulphate and carbonate, sawdust, flour, pulverized bone black, infusorial earth, etc. Where the amount of solids in the filtrate is small or where the solids are of such nature that they readily pass through ordinary filter media resort may be had to a filter aid to coat the medium and to prevent the filtrate from issuing from the press as a cloudy fluid. Filter aids are also used to increase the brilliancy of the filtrate and to assist in filtering solids which are adsorbed more or less readily by them. In such cases the aid after having adsorbed the solids or colloids is run with its liquor into the press with the same slight degree of trouble that is had with easily filterable materials.

Filter aids may be employed in three ways. First a slurry may be made up with water and be passed through the press ahead of the material to be filtered. Second, particularly adaptable where a strong adsorptive effect is desired, the aid is mixed with the material to be filtered and both are passed through the press. In the final method enough of the aid is mixed with the material to be filtered to give a coat to the medium. Experience will determine the best way to use the aid and quite a field of research remains to be explored to determine the best aids for particular substances and the proper method of employing them.

The earthy filter aids cannot be used with caustic or on some food products as they give an objectionable taste to them.

Kelly Filter.—This consists of a pressure tank mounted on a frame. A quick closing tank head is mounted on a carriage and track and this quick closing head and carriage support the filter leaves. The tank head contains all the connections for introducing washes, compressed air, etc. The axis of the apparatus is slightly inclined to allow the filter leaves and filter head to run down and out of the pressure tank when the cake is to be discharged. The cycle of operations with the filter which requires usually about an hour beginning after a cake has been disposed of, is: (1) To run the leaves and head into the pressure tank and close it. (2) Material to be filtered is then introduced under any desired pressure and after filtration has begun it is only run fast enough to maintain any desired pressure until the cake of desired thickness has been accumulated. (3) The material in the tank at the time the cake has accumulated to the proper thickness is returned to the supply tank. (4) Washing is done under pressure. (5) The press is opened, the leaves are run to the lower end of the frame and the cake is discharged by compressed air.

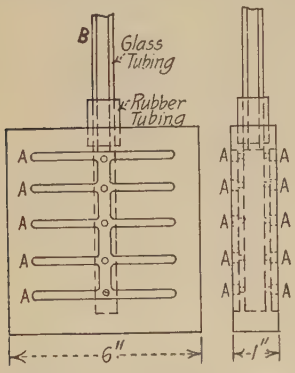

FIG. 25.—Gravity filter block.

Tanks can be built in a wide range of sizes. A tank 30 in. in diameter and 6 ft. long will have a net cubical capacity when the press is closed of about 25 ft. and a filter area of 140 sq. ft. A tank 6 ft. in diameter and 15 ft. long will have a net capacity of about 400 cu. ft. and a filter area of about 1,100 sq. ft. This represents about the standard range in size.[1]

Pressing.—Mechanically there are two principal types, the screw or hydraulic ram type and the worm type of press. The hydraulic presses are again divided into curb and plate presses. The curb presses consist of a cylindrical vertical bundle of slats or a cylinder of pierced metal which answers the same purpose, this is called the curb and is the material container. The liquid pressed out by the ram passes between the opening between the slats or through the holes in the metal cylinder. The curbs are hinged along a vertical line and open for removal of pressed material and for cleaning along a line facing the hinge. The slat curbs are held together with straps and are tightened after closing with cam locks. The curbs rest on the ram which is fashioned at its upper end into a saucer or gutter to receive the liquid which escape through the curb. The curb with its content of material to be pressed is forced up against a circular plate which fits closely

[1] For experimental work on processes where a filter press will probably be used in the final process and a small experimental filter press is not available, the effect may be gained by the use of a filter block as shown in the illustration. A wooden block from 6 to 14 in. square and 1 to 1½ in. thick is used. Grooves, *A*, are cut on the surface which connect with one long groove and thence by small holes, ⅛ in. to a central hole. A glass tube is placed in this, held by a short rubber tube to act as washer. The block is covered with filter cloth. A long piece of rubber tubing is then slipped over *B* and the whole then used either as a siphon, or connected with an aspirator and used with artificial vacuum. For large drain pits and for the bottoms of filter tanks the socalled "Filtros" blocks are valuable. For experimental work and even for small scale factory work, the Gooch crucible can often be profitably imitated on a large scale. I have made them 40 in. in diameter, with ⅛ in. holes in the board bottoms with filter cloth as the separating medium. See also p. 867.—EDITOR.

within the curb and which forms the upper or fixed pressing surface the ram with the curb mounted upon it providing the moving pressing means. Presses of this kind are much used for separating oil from lard, scrap and tallow.

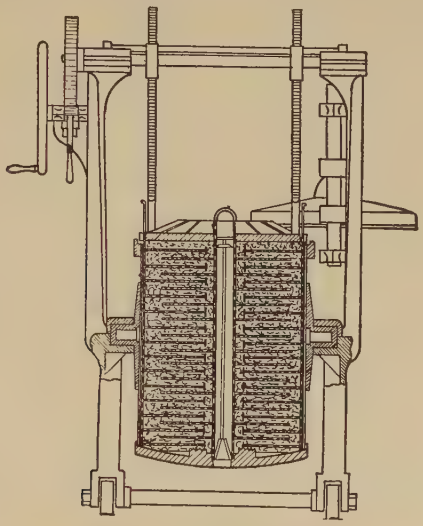

FIG. 26.—Brewery press.

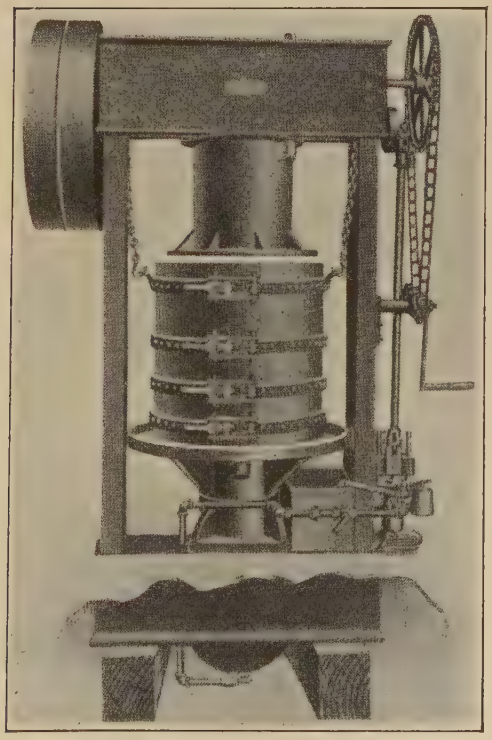

FIG. 27.—Hydraulic curb press.

Hydraulic plate presses have similar general arrangements the greatest difference being that the curb is replaced by a battery of plates. Presses of this kind are used by packing houses for extracting oleo from fats in the cold. A horizontal type is used for hot stearic and oleic separations. Power plate presses are also used for expressing seeds and nuts. In pressing cotton seed, the meats after hulling the bolls and after being cooked for 20 to 30 minutes are pressed between the plates of a hydraulic press. The meats are placed in hair cloth sacks, there being one to each plate. A pressure of 3000 to 4000 lb. per sq. in. is required and the solid material left in the sacks becomes as hard as a board under the pressing action.

Worm presses have horizontal shafts and consist of a powerful conical worm surrounded by a conical curb. The material is fed in at the large end of the worm and is advanced by it to the small end where it is discharged. During its progression the material is gradually compressed by the taper of the worm and the liquid squeezed out passes through the opening in the slats of the curb. Presses of this kind are used in the reclamation of old rubber. They can be advantageously used on some materials which have been washed to remove foreign matter or which have been digested with steam to liberate the grease and oil they contain but which still retain much liquid after a thorough draining. If the worm be mounted on a fixed hollow shaft with pierced holes the press can be used to remove oil, grease and other substance by the use of steam introduced through the hollow shaft.

Separation by Centrifugal Force.—The three possible separations which can be made by centrifugal machines are: (1) The extraction of liquids from solid material, such as textiles, fibers, leather, crystals, etc. Examples of this mode

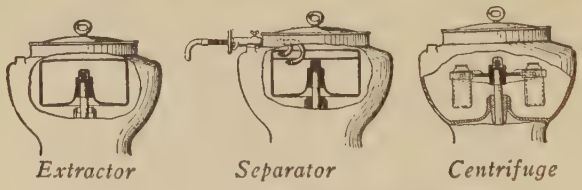

Fig. 28.—Laboratory convertible centrifuge.

of separation are to be seen in dyeing and extraction of excess dye and other liquids in dyeing. Extraction of molasses from raw sugar or muscovado and washing and coloring cane and beet sugar and the numerous centrifuging operations with massecuite in refining sugar. (2) Separation of one part of a liquid or emulsion from the balance. In adapting centrifugal machines to this end the basket of the extractor type is replaced by a curved bowl. Under rapid rotation the heavier part of the liquid will take a position next to the inner surface of the bowl and the lighter will occupy an inner ring of liquid and by a pair of pipes at the top of the bowl, one for each of the two liquid rings formed and dipping into them continuous removal of the separated liquids may be effected. (3) In the operation called "centrifuging" the material to be separated is introduced into swivelled bottles or the swivelled receptacles affixed to a rotating shaft. On rotating, the receptacles will swing out to a straight position and the material placed in them will arrange themselves in zones, the heaviest being at the bottom of the receptacle and the lightest near the mouth. Centrifuges are much used in the laboratory for the quick estimation of volumes and weights and the test receptacles used are often graduated so that percentages of volume or weight can be read off directly. Figure 28 of a convertible laboratory machine also illustrates dia-

grammatically the three modes of employing centrifugal force in centrifugal machines.

Separations Suitable to Centrifugal Force.—The use of centrifugal machines, owing to their cost and relatively low capacity, should be limited as far as possible to finishing operations. They are suited to drying operations where heat would destroy or injure the material to be dried or where in would be dangerous as with nitro-cellulose. In a very general way the greater the bulk of the individual unit of the solids introduced into a centrifugal machine the better it is suited to centrifugal separation. Also in a general way the less the percentage of liquid absorbed by the material to be separated the more suited it is to centrifugal separation. Where the liquid is not absorbed by the solid as oil on metal chips, shavings and small machined castings, the centrifugal separation offers a ready and expeditious and often the only means of removing the liquid. Where the liquid is not absorbed by or held by the solids as mother liquor in separating the crystals from it comparatively large proportions of liquid are permissible in centrifugal separation. One of the most extensive uses to which centrifugal machines may be put is in getting rid of pore moisture and chemicals in liquid form in textiles, leather, nitro-cellulose, etc., following washing, dyeing or vat treatment of some sort. When other separation means would be out of the question or too prolonged as in separating sticky viscid liquids from solids such as molasses from muscovado the centrifugal separator effects a removal which is rapid and complete. Finely divided amorphous material cannot be separated in a centrifugal machine used as an extractor or filter. In such cases where the conditions are proper for it the bowl can be substituted for the basket, the solids adhering to the inner surface of the former being removed after they form coats of the desired thickness.

The principal advantage of the centrifugal machine lies when it can be used as an extractor as the solids can be completely freed of liquid in one operation.

Details of Centrifugal Extractor. Fig. 29

Size of basket	30 in.	36 in.	42 in.	48 in.	54 in.	60 in.
Revolutions per minute	1,150	950	800	700	600	500
Velocity in feet per minute	9,040	8,954	8,796	8,796	8,483	7,854
Centrifugal force of 1 lb. at periphery, pounds	563	461	382	334	276	213
Approximate shipping weight	2,200	2,700	4,000	5,000	6,200	7,600
Horsepower of motors Type "E."	3	3	5	7½	7½	10

Details of Centrifugal Separators and Use.—No continuously operating separator of the extractor type has been successful. The stumbling block to attaining this end has been the continuous removal of the solids formed but a choice of machine best suited to the work will often increase capacity. If there be not much liquid to be extracted the batch to be placed in the machine may be worked in a removable basket.

Hydro extractors with top drives are the most common kind. The drive may be by motor, direct-connected steam engine or by belting. Figure 29 shows a motor-driven machine. Balancing of this machine is done through three swivel bolts connected to the stand. It is necessary to have the center of gravity of the basket and its contents as nearly invariable in position as possible otherwise destructive stresses

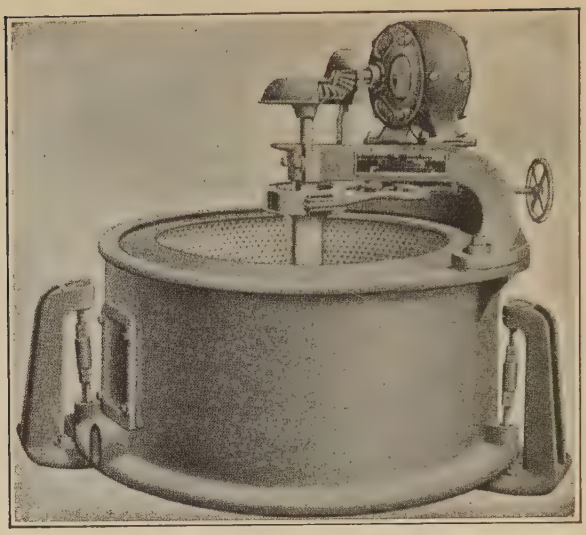

FIG. 29.—Motor-driven extractor.

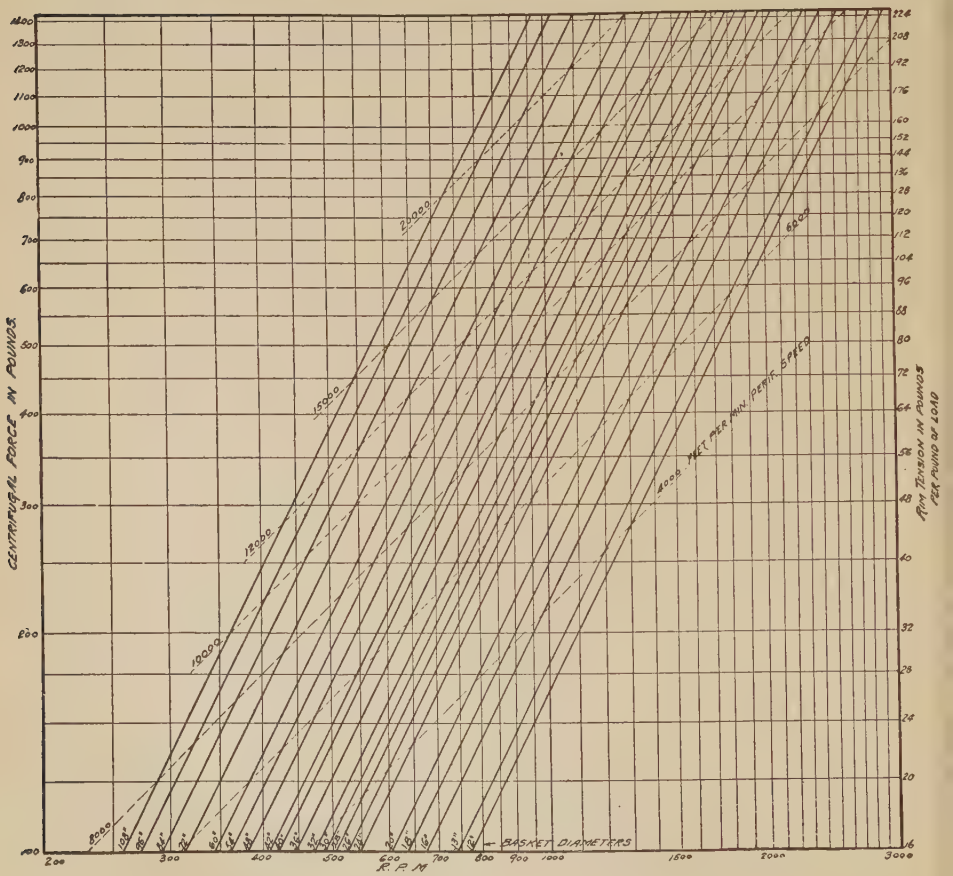

FIG. 30.—Centrifugal forces at various speeds.

will be set up. Some form of flexible mounting is necessary. Top-drive extractors are well-suited to textiles except when they are in the piece. For piece goods an under-driven goods machine is preferable for ease in loading and unloading. The under-

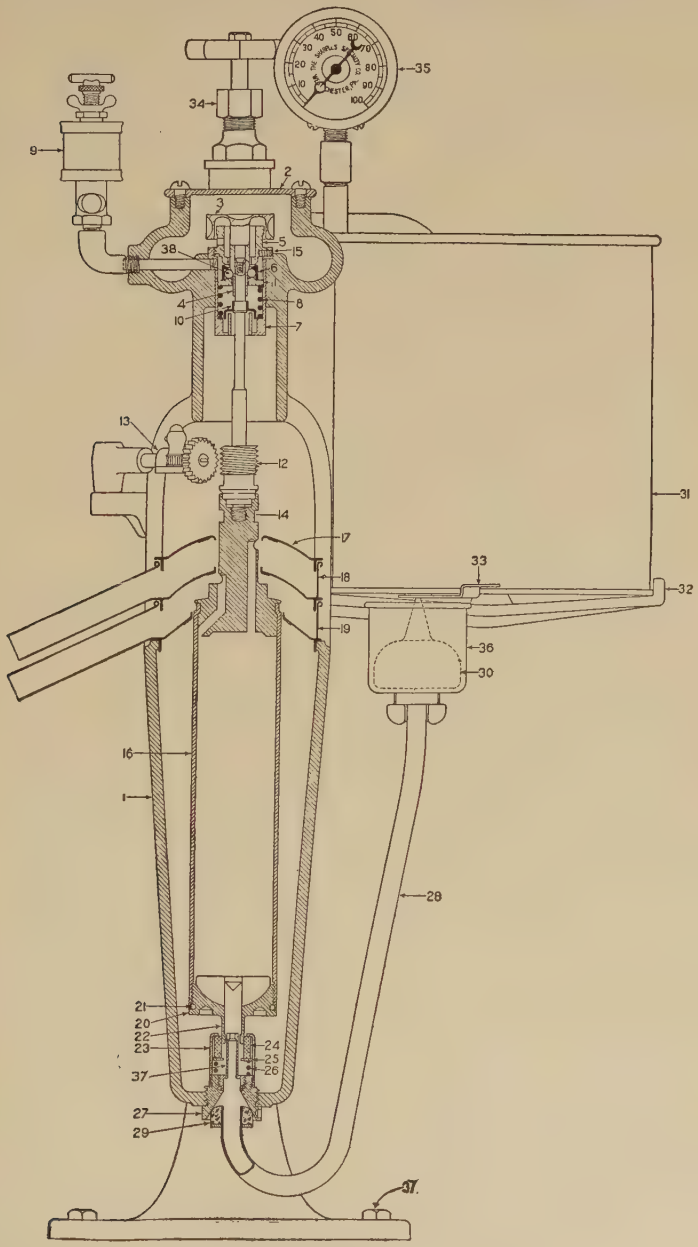

FIG. 31.—Sharples centrifuge.

driven machine is also required when batches of material are brought to the separators in removable baskets.

Bottom discharge is often the most convenient way of disposing of the solids particularly if they will slide through freely like coffee, small castings, etc.

The accompanying chart(p. 308) shows the centrifugal force which will be obtained with baskets of varying diameter and with the ordinary ranges of revolution.

The Sharples Centrifuge.—In this machine the bowl revolves from 20,000 to 40,000 r.p.m. and an effect is secured in separating solids, emulsions and matter in the colloidal state which is out of the range of the ordinary centrifugal separator. The formula for centrifugal force as given in the textbooks is F equals Wv^2/gr. If for v^2 is substituted its equivalent $4\pi^2r^2N^2$ and replacing the radius r by the diameter D, F becomes equal to $2W\pi^2DN^2/g$. Increasing the number of revolutions, N, increases the centrifugal force very much faster than increasing the diameter of the bowl as the centrifugal force increases as the square of the number of revolutions and only directly as the diameter. As in boiler problems the pressure tending to burst the bowl is equal to the centrifugal force expressed in pounds per square inch times the area of the section passing through the center of the bowl divided by two. If the internal diameter of the bowl is 30 in. and its depth 20 in. the area of the section on which the bursting pressure is figured will be 600 sq. in. It is evident then that other things being equal that high centrifugal force can be most readily obtained by bowls of small diameter revolving at high speed and it is the application of these two principles which is employed in the Sharples centrifuge.

Figure 31 shows a section of the separator for making liquid separations there being two discharge spouts as the figure will show. These machines are rotated by a steam turbine. The means of suspension, resistance to shock and of enabling the bowl and contents to maintain the position of its center of gravity constant in position will be evident from the diagram. See p. 321 for detailed list of parts.

The size of the machines range from a bowl diameter of 1¾ in. and with a depth of 9 in. which is a laboratory machine to a factory size with bowl diameter 4½ in. and a length of 36 in.

Some of the commercial uses which the machine suggests are the recovery of oil from soap stock, the dehydration of crude petroleum and emulsions, the separation of amorphous wax from cylinder stock and the recovery of wool grease from waste scouring liquors. A brief description of the last application will make the commercial use of the machine plain.

The figures are based on handling 30,000 lb. of wool a day with an average shrinkage of 60 per cent and using 4 lb. of water per pound of wool. As much as 14 lb. of water to the pound of wool are used in scouring but with centrifuging in view the amount used should be as small as possible and with less water there is a great saving in soap effected. Each No. 6 Sharples will handle 120 gal. of scouring liquor per hour and for 120,000 gal. of liquid 14 machines will be required and one dehydrating centrifuge for the grease from the primary machines for a 10-hr. day. These 15 machines will cost $15,000 and with extra parts the cost will run to $16,000. The manufacturers estimate the cost for power for the centrifuges will be about 2 cts. per hour per machine and that the yearly cost of operating the centrifugal plant will be about $10,000 including interest on the investment and depreciation.

The greasy liquors from the scouring bowls are first run to settling tanks so as to settle out sand and dirt and are kept at a temperature of 160°F. They are then run by gravity to the primary centrifuges from which grease containing 30 per cent moisture can be discharged. From the centrifuges the watery grease is removed and dumped into a grease tank which is equipped with steam coil and agitator. To the recovered

grease fresh water is added and the temperature brought to 160°F. It then passes to the secondary centrifuge where the last trace of water is removed. The scouring liquors will average about 1 lb. of grease to 80 lb. of liquor treated, there is consequently recovered about 1,500 lb. of grease per day of 10 hr. or for a 300-day year 450,000 lb. which at 14 cts. a pound is worth $63,000.

Some of the laboratory separations which may be effected and some suggestive of possible commercial separations are: bacteria from water, emulsions and serums, pigment from paint, fractional separation of clay particles, clarification of fruit juices, clarification of varnishes and glues, and the separation of emulsified oils from an unemulsified portion.

SEPARATION OF SOLIDS FROM GASES

Dust Chambers.—In theory, with dust laden air at ordinary temperatures a dust chamber of large cross-section and relatively short length should be as effective as a long chamber of small cross-section. With hot fumes the long chamber is preferable because of lowering of temperature with consequent reduction in volume and velocity and with increase of moisture in the non-gaseous form. Reduction of temperature on the other hand tends to increase the adsorption of the solid particles and increases the density of the air but these factors in decreasing the rate of settlement are not of much moment. If the fumes are hot but non-corrosive the wall of the settlement chambers should be as thin as possible. In one zinc-oxide dust-collection system the dust-laden air is cooled by passing it through pipes of thin copper. The exigencies of fume collection in smelting are such that only massive chambers of brick work are found to be satisfactory. In lead-smelting operations it is necessary to have fairly high velocities in the dust chambers in order to maintain the pressure sufficient to force the fume through the filtering bags of the baghouse. In this case settlement of dust in the chambers is not so important as collection of fume and it may be generally stated that when the particles are too small to settle in any dust chamber and require a filtration or positive means of separation after dust-chamber passage the extent of the dust chamber becomes of secondary importance. In the case of lead smelting it is the great volume of the dust and air and its high temperature which requires extensive settlement chambers. Much material is obtained by settlement but the most valuable material in point of assay comes from the baghouse. If the dust which is collected comes from cold currents and only the very fine particles are of commercial value then a settlement chamber of limited cubical contents is in order. If the volume of gases is small and it is merely desired to get rid of a dust nuisance, dust chambers need not be installed and the separatory arrangements will resolve themselves into some form of separatory or knock down devices.

Tortuous chambers and chambers arranged with baffles should be avoided as they tend to back up the draft.[1] It must be evident that in order to move the dust and fume either by artificial draft or by natural draft there must be a certain velocity and pressure at the point where the dust originates. If obstruction be employed in the dust chamber the pressure and velocity at the point where the dust originates may be

[1] Much better than baffles are plates hung with axes parallel to the flow of the gas. Wires suspended in the chamber are also more valuable than are the transverse baffles, but after all it is questionable if anything is an improvement upon making the dust chamber large enough to slow down the gas current to less than 5 ft. per sec., and for true fume nothing can replace the bag house or Cottrell installation.—EDITOR.

so small that the dust particles will not rise properly. If there be a fan and on one side it is pulling and on the other pressing against the resistances necessary in filtration or other separation means there must either be a serious loss in pressure or an unnecessarily large fan must be employed. All this is avoided by merely having a dust chamber of ample cross-section and as free as possible from bends and turns. The power losses in bends and contractions are well analyzed by Iles.[1]

Where only the finest part of the dust is of commercial value a tortuous or constricted chamber will create vortices and there will be settlement by centrifugal force with removal of fines at the wrong point.

The best form of dust chamber is one of comparatively great length compared with the other two dimensions and it has the additional advantage that hoppers can be more cheaply placed in the bottom and with less loss of head room. Also since the bulk of the material which will settle promptly will be found near the entry of the chamber less material has to be discharged in the long chamber as the dust at the far end needs only occasional removal. There is also a more practical gradation in size of particle from entry to exit end.[2]

Water in Dust Settlement.—In nearly every case where water can be used it has a beneficial effect. It is of much assistance in the Cottrell process of dust settlement to be described later. The more finely divided the water, the less need be the amount used but that the effect is somewhat independent of the quantity, that is that a large quantity in bulk will do effective work is shown by the experiments of Iles.[3] Jets of water were allowed to flow down through a tin box with slightly inclined alternate shelves so as to give the water a tortuous course. From 90 to 95 per cent of the smoke and fume was collected from the gas ascending the box.[4] Humidifying can be accomplished by nozzles or by centrifugal apparatus, which whirls the water into an impalpable mist.

Separation by Centrifugal Force.—The "Draco" separator is illustrated in Fig. 32. The figure also illustrates a water attachment which has much merit on material which will not be injured by water. Separators resembling the dry portion of this separator are manufactured by all the makers of grain cleaning machinery. The water attachment which gives atomized water by a series of spraying nozzles cannot be used successfully on materials which have no affinity for water such as coal dust, graphite dust, soapstone. One of the suggested uses for it is hydrating the dust which arise from lime hydrators.

[1] "Lead Smelting," p. 146 *et seq.*, John Wiley & Sons, New York, 1902.

[2] It has been proposed to build whole dust chambers out of extremely porous brick and let the chamber have no vent except through the porous walls. The difficulty appeared to be that eventually the pores fill up and cannot be freed.—Editor.

[3] *Op. cit.*

[4] Whether these results can be duplicated on a larger scale is open to some question. The experience of silver refineries with scrubbers where the water was not atomized has been that much valuable material goes by them. Despite its highly hygroscopic character, phosphorus pentoxide can be carried in a current of air through a number of wash bottles. In several scrubbers in which a multiplicity of 4-in. pipes carried the gases from a smoke-box into which they were forced by pressure down through water (the gases bubbling from the end of the pipes) the recovery was extremely poor. According to Iles a weak vinegar solution would have been more effective than pure water. A good experimental scrubber that I once saw consisted of a flattened cone containing a multiplicity of small holes and immersed in water. The gas was delivered at the top of the cone under pressure and then passed out the small holes into the water. Scarcely a trace of fume passed it, but this was a small installation handing only about 75 cu. ft. of gas per minute. That the ordinary scrubber is ineffective can be shown by drawing a known quantity of the effluent gas through absorbent cotton. The teachings of the colloidal chemists would lead to the belief that no scrubber could be successful except where the water was highly subdivided, and the success of the Feld scrubber is a brilliant testimony to the soundness of their theories.—Editor.

This type of separator is extensively used with grain separators and for drying and separating lint, etc. It will not separate the very finest particles and it is best suited to separations where the volume of air to be handled is

Number	Wet separator		Capacity, cubic feet per minute	Dry separator		
	Dimensions	Water, gallons per minute		Dimensions in inches		
	A			*B*	*C*	*D*
1	66	2½	1,100	66	35	6½
2	75	3	1,500	84	43	8
3	84	3½	2,300	100	51	10
4	98	4	3,600	115	59	12½
5	104	5	4,400	131	67	14½
6	109	5½	5,900	150	75	17
7	117	6	6,900	166	84	19½
8	124	6½	8,200	173	92	21½
9	130	7	10,000	178	100	24
10	135	7½	11,800	197	107	26
11	146	8	13,600	204	115	28
12	156	8½	15,500	220	124	29½
13	166	9	17,900	230	132	31
14	174	10	20,300	247	140	32½
15	189	12	23,100	253	148	34
16	200	14	26,500	272	156	36

small. In the Raymond pulverizing and separating system by close circuiting the exit of the centrifugal separator with the grinding devices used the escape of fine particles is avoided and taken care of in a simple way. The Raymond system with its so called four-roller high-side mill has a capacity on talc of:

1 ton per hour 98 per cent through a 300-mesh screen.

2 tons per hour 99.5 per cent through a 200-mesh screen.

3 tons per hour 98 per cent through a 200-mesh screen.

A wet scrubber that will handle fume as well as dust is the Feld illustrated in Fig. 33.[1] This consists of a number of separate washing chambers, one above the other. The openings in the bottom of each chamber act as passage for the ascending gas as well as overflows for the counter-current flow of the absorbing liquid.

The vertical shaft suspended from a self-centering, anti-friction bearing is provided in each chamber with a series of concentric frustums of cones. The spacing between the bottom of the chamber and the bottom of the cones increases toward the shell, to provide for their equable immersion in the scrubbing liquid, as well as to secure adequate agitation to avoid deposition of solids.

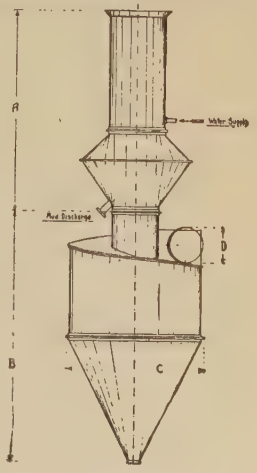

FIG. 32.

[1] From article by W. F. Lamoreaux, *Eng. and Min. Journ.*, Feb. 4, 1922.

When the vertical shaft revolves, these cones pick up or pump the scrubbing medium, which is thrown horizontally with considerable velocity through the outer perforated distributing cone. Additional impetus is given the liquid at this point

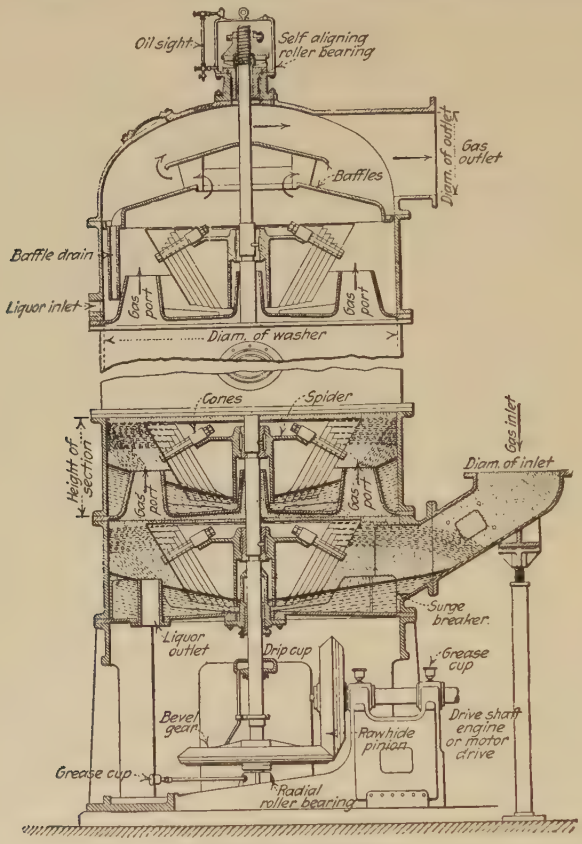

FIG. 33.—Feld Washer.

DIMENSIONS, WEIGHTS AND CAPACITIES

No.	Filter area, square feet	Overall dimensions				Outlet *d* diameter	Intake *e* diameter	Capacities				Approximate weight
		A	*B*	*C*	*D*			7.5 cu. ft. per square foot	10 cu. ft. per square foot	12.5 cu. ft. per square foot	15 cu. ft. per square foot	
2	384	124″	4′-0″	3′-8″	3′-6½″	15″	15″	2,980	3,840	4,800	5,750	3,000
3	576	128″	4′-0″	3′-8″	3′-6½″	18″	18″	4,300	5,760	7,200	8,650	4,500
4	768	129″	4′-0″	3′-8″	3′-6½″	20″	21″	5,750	7,680	9,600	11,500	6,000
5	960	130″	4′-0″	3′-8″	3′-6½″	22½″	23″	7,200	9,600	12,000	14,300	7,500
6	1,152	132″	4′-0″	3′-8″	3′-6½″	24½″	25″	8,700	11,520	14,400	17,300	9,000
7	1,344	132″	4′-0″	3′-8″	3′-6½″	26½″	27″	10,200	13,440	16,800	20,100	10,500
8	1,536	134″	4′-0″	3′-8″	3′-6½″	28½″	29″	11,500	15,360	19,150	23,000	12,000
9	1,728	134″	4′-0″	3′-8″	3′-6½″	30″	30½″	13,000	17,280	21,600	26,000	13,500
10	1,920	136″	4′-0″	3′ 8″	3′ 6½″	31½″	32″	14,400	19,200	24,000	28,800	15,000

by the greater rim speed of the distributor, and the liquid is projected through the gas space in the form of a fine, high-velocity spray, which, upon impact with the shell, is broken into a mist or fog. This alternating humidifying and scrubbing effect to which the gas is subjected in each chamber provides the intimacy of contact between gas and liquid necessary for the removal of finely divided insoluble matter or for the recovery of soluble constituents existent in extreme dilution, and also provides for the almost universal application of the apparatus as a scrubber as well as a reaction or absorption tower.

The vigorous liquid agitation eliminates the possibility of internal stoppages, as has been demonstrated on scrubbers operating for several years using liquids carrying suspended, insoluble, reactive material. Any liquid entrainment is removed from the gas by means of the deflectors or baffles in the dome, and the collected liquid is returned through the drain to the top chamber.

For primary washing, the Feld scrubber is ordinarily constructed with seven chambers or sections; the lower three being the washing chambers, the fourth one being a separating chamber and the upper three being the cooling chambers. The washing is accomplished mostly in the lower sections, whereas the upper sections perform primarily the function of cooling and dehumidifying the gas, all of the functions being subject to variation at will, depending upon the amount of gas passed through the scrubber and upon the volume and temperature of the scrubbing liquid used.

A Feld gas scrubber of standard design 5-ft. diameter, is estimated to be capable of cleaning 200,000 to 250,000 cu. ft. of gas per hour and is driven by a 10-hp. motor.

The volumes of flow of both the gas and the liquor are at all times under control and can be varied and relatively proportioned to meet operating conditions within the rated capacity of the scrubber.

It may readily be shown by experiment that when drops of water are broken up (as in falling raindrops, or by atomizing water by means of spray nozzles), the particles become positively charged with electricity. The positively charged particles condense more readily on negatively charged particles or on negative ions than they do on positively charged particles, and, also, liquid drops take up negatively charged particles or ions more readily than positive ones.

When dust is blown about by the wind, it becomes charged electrically, and from experiments made by Rudge, the following generalizations were deduced:

1. Non-metallic elements give *positively* charged clouds when the finely divided solid material is blown into a cloud by a current of air.
2. In similar manner, metallic elements give *negatively* charged clouds.
3. Solid acid-forming oxides give *positively* charged clouds.
4. Basic oxides give *negatively* charged clouds.

In the treatment of material containing or constituting salts, the charge apparently depends on the relative strength of adsorption of positive and negative ions.

Finely divided metallic sulphides generally acquire a positive charge, but the metallic oxides, sulphates, and sulphites acquire negative charges. In general, it has been found that highly dispersed substances, suspended either in gases or liquids, carry electrical charges, and this is particularly true of colloidal and ionized substances. It may be stated that, as a rule, basic colloids acquire negative charges, whereas acid colloids acquire positive charges.

The forces effective in the operation of a Feld scrubber then are: The spray or mist particles produced in the scrubber are *positively* charged, whereas the particles of fume contained in the smelter gases are *negatively* charged, and, all the physical conditions being such that these oppositely charged particles are brought into most intimate contact, the result is that these particles are mutually attracted and the fume particles are wetted and submerged within the liquid and thus removed from the gas stream.

Baghouses.—Woolen or cotton cloth bags of the general dimensions of 30 ft. in length and 18 in. in diameter are used. Wool is preferable to cotton because it resists heat and acids better. In a severe test by Iles with roaster smoke a test cotton bag had holes in it in 20 min. A woolen bag was eaten through in 2½ hr. At comparatively low temperatures both cotton and wool lose strength. Alexander has tested for the loss of strength in both cloths following the subjection of test pieces to varying temperatures and different amounts of time with the following results:[1]

COTTON: PERCENTAGE OF STRENGTH LOSS

Temperature, degrees Fahrenheit	Time in test oven, hours				
	1	24	48	96	144
210	...	0.00	...	...	0.0
215	...	0.00	0.9	6.1	9.4
220	...	4.20			
225	...	5.25			
230	...	7.30			
235	...	12.10			
240	...	17.10			
245	...	20.70			
260	0.3				
265	3.5				
270	5.3				
280	9.6				

WOOL: PERCENTAGE OF STRENGTH LOSS

270	...	0.0	0.0	0.0	0.0
275	...	...	...	0.3	3.2
280	...	...	0.0	5.3	6.9
285	...	...	0.0		
290	...	0.0	2.8		
295	...	4.0			
300	...	10.7			
355	0.0				
365	0.0				
370	2.3				
375	4.4				
385	6.7				

The nap which is on the cloth seems essential to the filtration process as cloth woven of material without nap lets through the fume. About 2 cu. ft. of fume per minute to the square foot of cloth surface seems to be the capacity of baghouse cloth, with 24 threads per in. in warp and woof. The bags are shaken by hand from one to eight times a day. If the bags are in bad condition owing to fume attack the slight shaking which this implies as compared with machine shaken filters tends to lengthen their life. For ordinary lead blast-furnace work cotton cloth is preferable on account of lesser cost. For corrosive fume wool should be favored and it is said that wool dyed with titanium chloride resists chemical attack well. The pressure at the bags is frequently less than 0.10 oz., more would be desirable. If the bags are well distended yet capable at the same time of being closed off entirely by pressure with the hands it is reckoned sufficient for the purpose of effecting filtration. At some of the western lead smelters bags are now shaken by machinery at periods ranging from 15 to 30 minutes. Average data with woolen cloth filtration at these smelters is: lb. of dust per cu. ft. of smoke, 0.00016; volume of smoke filtered per sq. ft. of bag per min. is 0.477 cu. ft. at 150°F.; sq. ft. of bag per cu. ft. of smoke 2.1; temperature of smoke entering bag house 150°F.; velocity in dust chambers 8.8 ft. per sec. (This is somewhat at variance with the dictum

[1] H. H. ALEXANDER, "The Baghouse in Lead Smelting," *Trans. A. I. M. E.*, XLIX, p. 561.

that the velocity of *fume* in dust chambers shall not exceed 5 feet per second.) Draft in distributing flue of bag house, 1.11 in. of water. Pressure in bag house pits, 0.98 in. of water. Zinc fumes are filtered through cloth but it requires a closely woven one. Zinc-oxide fumes after proper cooling have little or no chemical effect on the filter cloth and the latter will last a long time. Woolen bags are preferred for this work.

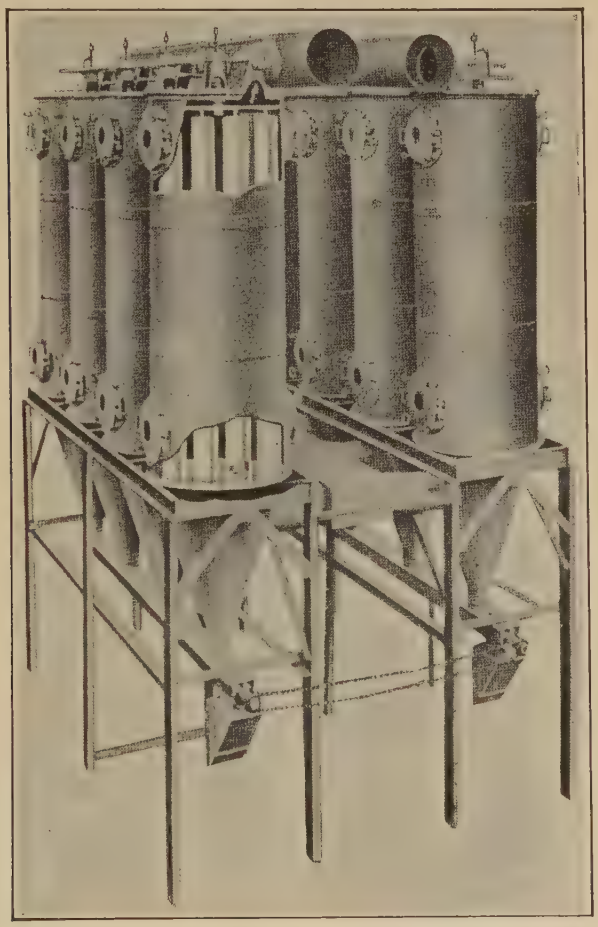

FIG. 34.—Perfecto mechanically-operated bag filter.

Mechanically Operated Filter Bags.—Figure 34 shows a generalized view of the "Perfecto" filter. An accompanying table gives dimensions and capacities, these being reckoned for a single cylinder which forms the unit. There are usually 12 bags to the cylinder. There is a timing gear which automatically permits the flow of compressed air to each cylinder for a period of about 4 sec. When the compressed air flows towards the cylinder the piston controlling the reversing gate acts immediately. When the gate is shut the dust-laden air is cut off, the cylinder is open to the outside air and the dust-laden

current exerts a suction on the bags. During the period of draft reversal the bags are vigorously shaken by a small compressor located at the top of the cylinder. It is possible to adjust the shaking apparatus so that the 4-sec. shaking period will come in 2 min. and from this up to 2 hr.

It is the great number of shakings and their thoroughness which gives this type of filter its capacity. On average gases it has a capacity of 10 cu. ft. of dust-laden air per minute to the square foot of cloth area. Closely woven cloth can be used because the vigorous shaking keeps the interstices open.

As an example of the use of the filter a zinc oxide separation can be described. The fumes arose from gas-fired reverberatories and were cooled by passage through 1,200 ft. of 20-in. copper pipe. About as much dust was collected in the cooler as in the filter. The filter plant consisted of eight unit cylinders each compartment con-

Stocking Dust Collector Dimensions

Number	Size of inlet	Size over all			Feet of cloth
		Length	Width	Height	
13	4¾ in. by 31 in.	4 ft. 6 in.	4 ft. 11 in.	6 ft. 6 in.	243
14	4¾ in. by 38½ in.	5 ft. 1½ in.	4 ft. 11 in.	6 ft. 6 in.	301
15	4¾ in. by 46 in.	5 ft. 9 in.	4 ft. 11 in.	6 ft. 6 in.	343
16	4¾ in. by 53½ in.	6 ft. 4½ in.	4 ft. 11 in.	6 ft. 6 in.	415
17	4¾ in. by 61 in.	7 ft. 0 in.	4 ft. 11 in.	6 ft. 6 in.	481
21	5¾ in. by 31 in.	4 ft. 6 in.	5 ft. 9 in.	7 ft. 6 in.	323
22	5¾ in. by 38½ in.	5 ft. 1½ in.	5 ft. 9 in.	7 ft. 6 in.	402
23	5¾ in. by 46 in.	5 ft. 9 in.	5 ft. 9 in.	7 ft. 6 in.	476
24	5¾ in. by 53½ in.	6 ft. 4½ in.	5 ft. 9 in.	7 ft. 6 in.	549
25	5¾ in. by 61 in.	7 ft. 0 in.	5 ft. 9 in.	7 ft. 6 in.	629
26	5¾ in. by 68½ in.	7 ft. 7½ in.	5 ft. 9 in.	7 ft. 6 in.	723
27	5¾ in. by 76 in.	8 ft. 3 in.	5 ft. 9 in.	7 ft. 6 in.	813
31	6¾ in. by 53½ in.	6 ft. 4½ in.	6 ft. 9 in.	8 ft. 9 in.	739
32	6¾ in. by 61 in.	7 ft. 0 in.	6 ft. 9 in.	8 ft. 9 in.	829
33	6¾ in. by 68½ in.	7 ft. 7½ in.	6 ft. 9 in.	8 ft. 9 in.	965
34	6¾ in. by 76 in.	8 ft. 3 in.	6 ft. 9 in.	8 ft. 9 in.	1,073
35	6¾ in. by 83½ in.	8 ft. 10½ in.	6 ft. 9 in.	8 ft. 9 in.	1,180
36	6¾ in. by 91 in.	9 ft. 6 in.	6 ft. 9 in.	8 ft. 9 in.	1,288
45	7¾ in. by 83½ in.	8 ft. 10½ in.	7 ft. 3 in.	9 ft. 3 in.	1,398
46	7¾ in. by 91 in.	9 ft. 6 in.	7 ft. 3 in.	9 ft. 3 in.	1,514

taining 12 bags 9 ft. long by 7 in. diameter or 190 sq. ft. of filtering area. The exhaust fan used was a No. 9 Buffalo Forge running at about 1,100 r.p.m. and produced a vacuum of about 5 in. of water. To force the gases through the filter required a head of about 3 in. of water. The volume of gas arriving at the filter was 9,000 cu. ft. per minute consequently the capacity of the filter per square foot of cloth surface was 6 cu. ft. The gases contained a little SO_2 and a little SO_3. Woolen bags are used and last a year. Shaking is at 2-min. intervals and requires 16 cu. ft. of free air

compressed to 35 lb. per square inch. Repairs about the filter in 2 years time have been very slight. The power required to run the installation is estimated at 20 hp. and 6.7 lb. of zinc oxide are recovered per minute.

The entering dust-laden air has to make a turn before passing through the bags, this assists in dropping out any large particles which would tend to cut the bag fabric.

Stocking Dust Collectors.—These are manufactured by most of the manufacturers of grain-cleaning and flour-milling machinery. The general form of these machines is illustrated by Fig. 35 with a body made of galvanized iron. The dust-laden air enters at *A* and is first given a centrifugal whirl to separate out any coarse particles or abrasive particles tending to cut the cloth of the stockings. The latter are arranged around the center of the separator and are secured to a center rotating drum. When the stockings come over the hopper

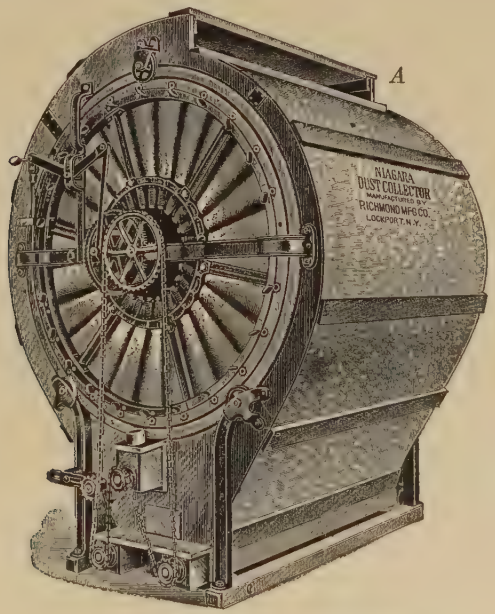

FIG. 35.—Stocking dust collector.

portion of the case means are provided for shaking and discharging them. For each square inch of fan opening the following number of square feet of filtering cloth is required: Receiving separators and corn cleaners 3 sq. ft.; milling separators 2½ sq. ft.; scourers 3 sq. ft.; oat clippers, 4 sq. ft., purifiers, 7 sq. ft. For roll suction for each double stand of four rolls, 6×12, 25 sq. ft. up to 50 sq. ft. for 10 by 36. The principal advantage of the stocking filter is that material dropping into the hopper may be withdrawn without interfering with the separation.

The Cottrell Process.—The Cottrell process is a separation by the silent or glow discharge from high-voltage direct current. The electrodes are usually a small pipe about 10 ft. long and 6 in. diam. through the center line of which is

suspended a fine wire forming the other electrode. The dust-laden air passes up through or occasionally down through the batteries of pipes forming one of the sets of electrodes and on coming within the influence of the high-tension current the dust particles are immediately more or less completely knocked down. The moment the particles are electrified they are attracted to the pipe or collecting electrode and by the use of an interrupter in the current circuit the dust particles will be able to work their way down into the hoppers into which the pipes are inserted. The Cottrell process is particularly valuable with fumes which would attack cotton or wool in cloth filtration and most of its first installations were around copper furnaces and converters the fumes from which containing large amounts of SO_3 could not be successfully filtered through cloth.

The great disadvantage of the process has been its large installation cost. It is estimated that the cost of installing a Cottrell dust-collecting system, together with leaching and evaporating systems for a byproduct potash plant from a cement mill of 1,000,000 bbl. annual capacity would cost about $150,000.[1] The usual range of K_2O in clay or shale suitable for portland cement is from 0.8 to 1.25 per cent. Amounts as high as 2½ per cent are occasionally found. A 100 ft. rotary kiln will produce about 4 to 7 tons of dust daily with potash content from 4 to 10 per cent. If soils require lime the collected dust may sometimes be used as a fertilizer without further preparation. Potash recovery from cement rests upon the observation of Dr. W. F. Hildebrand that after burning the raw mixture the greater portion of the potash and soda had been expelled by volatilization. Of the amount volatilized together with the dust from the kilns the Cottrell process will throw down from 50 to 80 per cent. The slow solution of the potash following the Cottrell process is due in part to the formation of an insoluble double sulphate of potassium and calcium. By heating the dissolving water to 85°C. and higher and maintaining these temperatures during the subsequent filtering operations the greater part of the soluble potash can be recovered.

In fume recovery[2] the Cottrell process gave 100 per cent precipitation from copper furnaces containing large amounts of SO_3. The proportion of SO_3 to SO_2 ranged from 1 : 18 to 1 : 8. It was necessary to reduce the temperature of the fumes to about 90°C. to get the best results. The velocity of the gases in the electrode pipes had to be less than 15 ft. per second. Water was also necessary to give good results, 3 to 5 per cent of the weight of dust collected. When there was much SO_3 the electrode pipes became non-conductors and water tended to overcome this difficulty. The fume recovered contained 41 per cent lead as basic sulphate.[3]

One curious result has been recorded by Howard: during the progress of the test work with fumes from various furnaces it was observed that during the blow on white metal, where a conductive deposit was obtained in the pipes practically all the lead was precipitated at a temperature of 340°C. but the SO_3 and the As_2O_3 escaped. To precipitate the latter and obtain complete smoke clearance it was necessary to reduce the temperature to about 90°C. At a certain western lead blast-furnace plant fumes

[1] *Cf.* "Potash in 1917, *Min. Res.* of the U. S., 1917," Part II; J. J. Porter, "The Recovery of Potash as a Byproduct in the Manufacture of Cement," presented at Fall Meeting of Portland Cement Association, Chicago, Sept., 1917, and J. Treanor, "The Experience of the Riverside Portland Cement Co. in the Recovery of Potash from Cement Flue Dust," presented at the December meeting, Portland Cement Association, 1917.

[2] *Cf.* "Electrical Fume Precipitation at Garfield," by W. H. Howard, *Trans. A. I. M. E.*, XLIX, p. 540.

[3] Spray nozzles of the Schütte-Koerting type can be used advantageously for moistening fume.—Editor.

from the furnaces are passed through 6-in. Cottrell pipes under a pressure of 0.22 in. of water. The fume per cu. ft. of gas is 0.000044 lb. and the velocity of the fume in the Cottrell pipes is 4.9 ft. per sec. The power consumed per 100,000 cu. ft. of gas at a temperature of 206°F. is 1.13 hp.

Developments in recent years with the Cottrell process have continued to show the greatest success with an acid atmosphere or conductor. Sulphuric acid mist can be readily removed and concentrated by the process. In the reduction of phosphate rock in electric or other furnaces, phosphoric acid is recovered by the Cottrell method and is used to saturate phosphate rock giving a superphosphate of high P_2O_5 content saving the use of sulphuric acid. The Cottrell process does not work with ordinary soot or ash. It has been said that the process has been worked out for the removing of the smoke from railroad round houses but no installation has as yet been put in operation. The engineers of the Research Corporation state they have had some success in such a basic problem as removing the lost carbon from burning natural gas in gas black production. There is now produced but 1 lb. out of thirty-three available by the present day methods.

The writer has never seen anywhere any statement as to whether as in cataphoresis certain sorts of particles tend to migrate to the positive pole and others to the negative pole and what effect this would have if the gases were basic or neutral. Nor has he seen any reference to experiments with artificial conducting atmospheres created by introducing with the solids to be separated various gases or vapors. But in this refer to the description of the Feld washer a few pages back.

One effect of the apparatus is to remove the gas adsorbed by the particles as in making water drops coalesce by removing the air envelope surrounding them by electrical discharge and thus permitting internal forces to have full play causing particles to come together, when they will settle more rapidly.

The Cottrell process will not throw down SO_2 and it apparently only throws down SO_3 through the formation and precipitation of sulphuric acid. Water helps on many problems. One use has already been mentioned. It has been stated that zinc fumes with the proper amount of water can be handled at temperatures as high as 600 to 700°F.[1] The water must be thoroughly mixed with the fume. Steam also helps in precipitating tin chloride.

The voltage maintained ranges from 30,000 to 100,000. Owing to difficulties in rectifying large amounts of currents the capacity of panel sets ranges from 5 to 50 kw. The Cottrell process can be worked at temperatures where cloth filtration would be out of the question.

PARTS OF THE SHARPLES CENTRIFUGAL.

(Numbers refer to figure on p. 309.)

1 Bowl Barrel	11 Ball Cup Plate
2 Steam Lid	12 Bowl Spindle
3 Steam Wheel	13 Speed Indicator
4 Cone Bearing	14 Bowl Neck
5 Ball Cup Cap	15 Ball Cup Lock Screw
6 Ball Cup Race	16 Bowl Shell
7 Ball Cup Shell	17 Cover Lid
8 Ball Cup Spring	18 Upper Cover
9 Sight Feed Oiler	19 Lower Cover
10 Oil Shedder	20 Bowl Bottom

[1] But most makers of zinc oxide believe that wetting fume intended for paint spoils it. Against this, however, is the practice of one maker, who washes and dries all of his oxide.—EDITOR.

21 Bowl Bottom Joint Ring
22 Bowl Boss
23 Drag Shell
24 Drag Bush
25 Drag Washer
26 Drag Spring
27 Feed Nozzle
28 Feed Tube
29 Feed Tube Rubber or Cork
30 Float
31 Supply Can
32 Supply Can Holder
33 Supply Can Faucet
34 Steam Valve
35 Pressure Gauge
36 Feed Tube Cup
37 Feed Nozzle Delivery Tube
38 Balls

SECTION VIII

CONCENTRATION

By Edward S. Wiard[1]

Methods.—The chief principles which are employed or taken advantage of in concentration are: (1) That the rate of fall of fragmental solids through liquids and gases is proportional to specific gravity and size. (2) The interstitial effect already explained under grading but for particles all of the same specific gravity. To employ the interstitial effect in concentration it must be accompanied by cross or back washing. (3) Washing down or back effect with thin beds of grains on plane or curved surfaces. (4) Magnetic concentration. (5) Electrostatic concentration. (6) Those employed under the generic term "Flotation."

These principles with the exception of numbers (4) and (5) are arranged in the order of coarseness of particles or fragments. Principle number (1) is employed from 2-in. sizes down to sand sizes. Of late years however the tendency has been to employ principle (2) on the sand sizes and from ⅛-in. size down to about 80-mesh size. This change came about with the advent of the Wilfley table which employs interstitial action with cross washing. Jigs are the chief machine for employing principle (1). They work best with screened feeds. Below ⅛-in. size it was formerly the custom in Western metal-mine ore-dressing plants to prepare the feeds for the sizes finer than ⅛-in. with rising currents of water. To operate jigs properly with classified feeds requires experienced labor and the advent of the Wilfley table where the operations are all visible and require little training for its attendants has caused the almost entire displacement of classifier fed jigs. Indeed the tendency today in Western mill practice is to do away with jigging altogether, principally on the score of simplicity and to do away with the wear and tear which screens and jigs imply and the skilled labor required for properly operating jigs and other and more important but not so evident factors to be discussed at a later point. Whether or no jigs should be a part of any mill plan will depend upon what the testing leading up to the mill arrangements which this discloses. If the tests indicate that any shipping product can be made by jigging then the use of these machines is certainly indicated as a possibility. In many cases even if jigging does not yield a shipping product it may often be used to advantage as a roughing machine. Where the material being concentrated is crushed to half inch or finer the use of some type of rougher other than jigs may be preferable.

The chief machine and devices employing principle (3) are the round and canvas table. For sulphide ores and material which is adapted to saving by methods of flotation the preference would be flotation rather than these devices if only one method is employed. The methods of flotation, while not the panacea for dressing problems that they were originally supposed, recover more in the size range to which principle (3) is applicable from about 80-mesh to the smallest which are visible to the eye. If desired flotation can precede machines employing this third principle or it can come after provided it is desired to recover an extra modicum which would not be yielded by flotation alone. Flotation like other methods is weak on the very finest material and while it is the practice to submit all the very finest material made to it provided it is of the kind which will "float" there are serious losses in the very finest sizes.

[1] Consulting engineer, 407 Boston Building, Denver, Colo.

Magnetic and electrostatic concentration are applicable to a range of size from about 12-mesh to the finest sands. They are seldom used as primary methods but in conjunction with the concentrations made by employing principle (2).

Unlocking.—Before tests can be made or concentration can begin the ore or other material to be dressed must be unlocked by crushing and comminution. The aim of crushing and comminuting operations should be to make the minimum of the finest material since this is the part most difficult to treat. It is usual to crush to a point where some commercial or waste material will issue on employing concentration operations and this first crushing may be termed preliminary crushing. The unfinished products which arise by concentrating the material which has undergone a preliminary crushing will need further reduction before being again submitted to the finishing concentrating operations. This recrush may be termed secondary crushing.

Preliminary Crushing.—Crushers, rolls and disc crushers are the principal machines employed in preliminary crushing. Rolls, edge runners, ball, rod and tube mills are employed in secondary crushing. But the general principles of crushing are extensively treated under "Crushing and Grinding" and will not be further referred to here except as necessary for this subject.

Principle of Secondary Reducing Machines.—In preliminary crushing machines the material to be crushed is caught between surfaces or crushing masses inclined to one another and is broken by the direct application of pressure. The impact machines for coarse crushing work are but little used in concentrating work. Owing to wear and lost motion it is impossible to extend the principle of coarse crushing into that of fine because the crushing faces could not be kept in pressive contact with the material to be crushed. The corrugations and irregularities of surface caused by wear will often prevent the crushing surfaces from being brought in intimate contact and fine material will escape through interstices formed this way. It is consequently necessary in designing secondary crushing machines to have a steel case enclosing the crushing faces or masses the ore being introduced inside of such case and more or less surrounding the crushing faces or masses. With edge runners such as chilean and Huntington mills the crushing masses are rotated by power. In the case of ball and tube mills the balls or pebbles rise by centrifugal force and friction until these forces are overcome by gravity. They then cascade upon the ore and balls or pebbles. In the case of stamps weights are raised by power and are then allowed to fall on the material to be crushed. Crushing in these cases is partly by impact and partly by attrition. In the case of the edge runners crushing is partly by pressure and partly by attrition. The difference in crushing principle between those applied in preliminary crushing and secondary crushing are that in the first case crushing is done by direct application of pressure the material being fed in a thin stream so that the greater bulk of the particles are crushed by being directly caught between the crushing faces. In secondary machines to overcome the impractibility of direct seizure of fragments between crushing faces, crushing where done by pressure or impact, which gives the same effect, is *en masse*, there being a relatively great number of fragments in the lines of the application of pressure.

In preliminary crushers the route of the material to be crushed from entry to exit point is as direct as possible. In secondary reducing machines the route is long so as to afford the fragments the opportunity to come under the influence of the haphazard

reducing agencies as frequently as possible. In preliminary crushers the crushing faces or masses are of no greater weight than is necessary to provide for wear and to resist the crushing forces. In theoretical power discussions all the moving parts of the reducing machines preliminary and secondary must be considered the crushing faces or masses. Bearing this in mind it may be stated in a general way that the weight of the crushing masses to the ton of ore crushed are inversely proportional to the size of the particle through the whole range of crushing and grinding machines. The crushing masses are least in weight in crushers and greatest in tube mills. Offsetting the inefficient application of power what this statement implies is that greater power is consumed in bearing friction in the cases of the preliminary machines. It must be evident that if there is reduction by direct application of pressure this pressure must be carried back to the bearings and result in great power consumption in the way of friction. In the case of stamps and ball and tube mills charged with their grinding balls or pebbles there is practically no greater power consumption between running empty and full. In these cases the power losses due to friction are caused directly by the weight of the moving or power masses. The power for useful work, that actually used for reduction, is a part of the power required to raise the stamp or balls or pebbles to the point where they fall on the material to be reduced.

The chief inferiority of the principles employed in secondary crushing lies in the fact that they lead to much power lost in uselessly moving about the material to be crushed. In a test made with Cripple Creek ores with a large chilean mill the power required for driving the mill empty was 9 hp. With a light feed the power increased to 28 hp. With heavy feeding to 63 hp.; and at the limit of loading the power required was 70. In the case of this grinding machine the bulk of the power used is employed in raising the wheels, and the height to which they must be raised increases with the load. At the same time as must be evident, great increases in power result from moving the greater masses of ore in the mill as the load increases.

Some authorities favor carrying roll crushing down to $\frac{1}{25}$ of an in. but the limit for these machines is today owing to other practical considerations about $\frac{1}{4}$ in. At one time there was and there is still an active school which advocates taking material directly from large crushers and feeding it to ball mills for reduction to sand sizes. This practice led to some disastrous deficiencies in expected tonnages. The tendency today is to consider material 1 in. in size and smaller as the maximum in point of size which should be fed to ball mills. As to whether coarser material can be fed with satisfactory capacities will depend upon its hardness.

Testing for Concentrating Processes.—The determination of the proper flowsheet for a difficult complex ore is a matter requiring not only careful testing but exercise of the nicest judgment. But refinements in treatment may be carried so far that a flowsheet while technically excellent may not be admirable from a commercial standpoint. That is, the increased saving which has been effected beyond a certain point may have been obtained at an expense which more than eats up the ore receipts from treatment refinements. The ideal ore for concentration is one where the mineral or other substance to be saved is in a pure state and arranged in masses or particles in the matrix or gangue. The larger such masses are the more amenable the ore is to concentrating methods. For gravitational methods the greater the difference in specific gravity between the mineral or material to be saved and the gangue the more readily it will yield to them. In the employment of the other principles employed in concentration which have been mentioned at the beginning of this section gravity is not the sole factor nor even the principal factor in concentration or an index of the ease with which concentration can be effected. Galena, sp. gr. 7.25 to 7.7, "floats"

readily but graphite, 2.09 to 2.229; molybdenite, 4.44 to 4.80; and gray copper, 4.50 to 5.56; also float readily as also do many organic compounds. The specific gravity of quartz and earthy materials and gangue ranges from 2.4 to not more, ordinarily, than 3.5, quartz being 2.67.

If more than one substance is to be concentrated out of a matrix or gangue the difficulties of separating one from the other and the gangue are very much greater than if a single substance is to be separated from gangue. If the plurality of substances to be concentrated is finely disseminated through the gangue or closely mixed with one another a concentration problem is presented which will either present insuperable difficulties in solution or lead to high costs for plant and operation. A microscope can often be used profitably here.

In testing for a flowsheet portions of the material to be treated are crushed to various sizes which an inspection before reduction would indicate as being the points where either (*a*) a sufficient quantity of concentrate can be obtained or (*b*) clean tailings can be produced. If clean tailings can be produced following preliminary crushing operations, great savings can be effected in the finishing treatment as the roughing concentration will reduce the quantity of middling material which has to be further ground before being submitted to the final concentrating operations. Hand-sorting tests should also be made to determine whether clean waste can be made. Concentrating by machinery used to be considered to begin in the neighborhood of 2 in. but 1 in. is today nearer the extreme upper limit at which concentration begins.

The best range for hand sorting is usually from 3 to 1 in. Above a piece of 3-in. size the difficulty of handling the pieces with despatch becomes increasingly difficult owing to their weight. Below 1-in. size the tonnage made per man by hand removal becomes too small ordinarily to make sorting operations profitable. Such small pieces are also difficultly recognizable. The successful employment of hand sorting largely depends upon the sorter being able to make an instant decision as to whether the fragments passing before him are ore or waste. In sorting the material of least tonnage should be removed. If there be but little ore in the stream passing the sorter the ore should be taken out. And this is the usual problem. Where there is only a little waste to be removed common sense will indicate that the sorting operation may be left out of the mill scheme.

The scheme of the flowsheet following the preliminary concentration consists in regrinding the middling products and submitting them to further concentrating operations. It will be evident that the tailings from the preliminary operation can be higher than those from the final operations. The amount of valuable substance which can be left go to waste in the tailings depends on the individual ore. It is best to err on the commercial side, that is, to leave in tailings more valuable substance than would be necessary if the most refined methods were employed. Each successive reduction and retreatment method is increasingly more expensive and at the same time the percentage recoveries diminish.

CONCENTRATION METHODS

Jigs and Jigging.—The principle of Archimedes which states that a body in a fluid loses a weight equal to the weight of the fluid displaced enables one to write a formula for the unbalanced force acting on a fragment of material subsiding in a fluid. The unbalanced force is $Vw(s' - s)$, where V is the volume of the grain, w the weight of a unit of water and s' and s the specific gravities respectively of the solid and fluid in terms of water. The acceleration of such a particle if there were no fluid resistance would be $g\frac{s' - s}{s'}$ and quite evidently if there

were no resistance the velocity a grain would attain after any period of time of subsidence would depend only on its specific gravity and would increase as the specific gravity increases. The size of the fragment is immaterial. When the opposition of the fluid is considered it is seen that a large grain will fall faster than a small one of the same kind and also that grains of high specific gravity will fall faster than ones of less specific gravity. What are called free settling ratios are those of relative sizes at which particles of different material will have the same settling rate. Free settling ratios are conventionally expressed in terms of quartz. The free settling ratio of galena is 4.01. This would be the same as saying that a grain of galena of 4.01 mm. diameter would settle at the same rate as a quartz grain of 16.08 diameter etc. If screens of these two sizes were employed to effect a grading or any pair having the ratio as multiplier or divisor all the grains of galena could be separated from all the grains of quartz by jigging.

Where there is a mass of ore in a fluid such as water the combined ore and water must be considered as a "fluid" and the specific gravity of this "fluid" must be used in making computations. This favors the settlement and separation of heavy grains smaller than those indicated by the free settling ratios. It will be evident further that since the heavier grains are hindered by their neighbors in taking a direct path to the screen, their average velocity in reaching the screen is very much less than their free settling velocities and hence the relative settlement rates will be more directly in proportion to specific gravity. Owing to this and other factors Richards by experimental work has determined that the true settling ratios are higher than the free settling ratios. He has determined for example that the ratio for galena to quartz is 5.842. He denominates these ratios of increased amount "hindered settling ratios." The accompanying curve shows the hindered settling ratios for common minerals. If two heavy materials are to be separated from one another as well as from quartz then the hindered settling ratio of the lighter of the two heavy minerals must be divided into that of the heavier to find the screen ratio, if the hindered settling ratio of the quartz and the lighter of the two heavy minerals is less than this ratio. For example the working screen ratio for a mixture of chalcocite, galena and quartz is not 5.842 or 3.115 but 1.875, the ratio of the hindered settling factors of the two heavy minerals.

The mode of operation of a fixed sieve jig with pure mineral fragments and screened within the limits imposed by hindered settling ratios may be gained from the following elementary description of the separation action. If two grains, Fig. 2, *G* and *M* of different volumes and different specific gravities, *M* being of greater specific gravity and either the same size or smaller within the limits imposed by hindered settling, be at equal distance above the screen of the jig as at points *A* and *A* and they be given an upward pulsion by a fluid such as water, then they ought to arrive at the end of the pulsion at some such position as *BB*, but since both fall to some extent and *M* falls faster than *G* at the end of the upstroke, *G* will be in some position such as *C*, and *M* as some position as *D*. On the downward return following the upward movement, both grains will be aided by their downward impelling force, but since this gives to *M* the greater velocity, it will take some such position as *E* and *G*, *F*. By a number of cycles as just described *M* will ultimately reach the screen, while *G* will still be at some distance above it. If now while falling relatively to one another, the two grains are subjected to a sidewise current, such as is produced by the flowage of ore and water through a jig, then evidently *G* will be more affected laterally than *M*, and may be caused to flow entirely away and to waste, while *M* though not reaching the screen in a position vertically under the point where it first enters the jigging compartment where the screen is, but ahead of it, can be drawn through the holes in the screen or

be removed by other means and a separation is thus effected by the first principle enunciated at the beginning of this section.

The knowledge of the resistance offered by a viscous fluid to the passage of a solid body through it, is far from being perfect. At high velocities, since there is a continu-

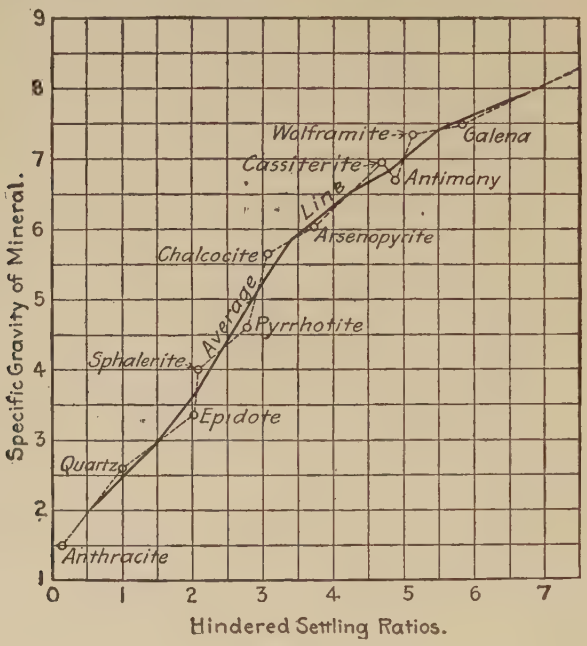

Fig. 1.

ous pushing aside of the fluid by a mass of cross-section A and with a velocity V it would be expected that the energy consumed would be all or a portion of the energy given out by a flowing stream of cross-section A and velocity V. In addition to this there would be a force required to overcome the viscosity of the fluid, and since the surface of the grain and the velocity with which it is moving are the origin of this viscosity, it would be expected that it would be proportional to both of these factors.

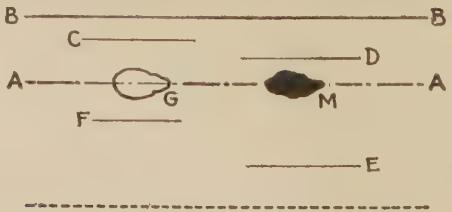

Fig. 2.—Theoretical settling in jigging.

Now at high velocity the retardation produced by viscosity is so small a portion of the total impedence that it can be neglected, and only the pressure produced by the moving grain need be considered. This is found to vary with the shape of the grain; for example, a smooth shuttle shaped grain, moving downward with its long axis parallel to the direction of motion offers no resistance or practically no resistance to the fluid, for the anterior pressure created by the down movement is balanced by a minus posterior pressure, and there exists no tendency whatever to set the fluid outside of such a body in motion. If, however, a flat grain be falling in a fluid, and

with its long axis at right angles to the direction of fall, then since its advance causes the fluid to flow in a multitude of radial lines across the anterior face of the grain, and this sets the fluid in motion beyond the edge of the grain, the anterior pressure is largely used up in producing motion and very little is recovered in minus posterior pressure. If the pressure which the moving grain exerts is equated with the expression for the unbalanced force which has already been given and the value of V is transposed there is obtained for water, sp. gr. 1, the expression for V of $2.67\ [D(s'-1)]^{\frac{1}{2}}$ for a sphere. Rittinger who investigated the fall law experimentally gives the value of V as $2.44\ [D(s'-1)]^{\frac{1}{2}}$ both the diameter D and the velocity V being in meters. These results are for grains of irregular shape. In the sphere formula the fall velocity is in feet and the diameter D in inches.

By referring to the curve mentioned on page 328 it will be seen that there is an inflection in it. This is the point where owing to slow rate of subsidence owing to reduction in size the resistance due to the viscosity of the fluid becomes of more moment than that offered by the fluid in being pushed aside. For small particles falling through fluids and gases the famous Stokes formula applies. Its application for air and gases has already been touched upon (p. 225). In deriving this formula the unbalanced force acting on the particle is the value which has already been derived in preceding paragraphs. The resistance of a small spherule is given by the expression $6\pi rkV$, where k is the coefficient of inner viscosity and r the radius of the spherule. The mathematical steps for obtaining this expression are very involved.[1] Written in the absolute system and with the proper transformations V becomes $2/9kr^2(s'-s)g$. This formula of course has no application in jigging as it has to do with fall velocities for particles very much below the lower range size for this operation. Richards has found that the critical velocity or point below which the resistance tends to vary directly as the velocity rather than as the square of the velocity is to be found for galena and quartz at 0.13 mm. diameter and 63 mm. velocity per second for the former and 0.20 mm. diameter and 28 mm. velocity for the quartz.

Types of Jigs.—Jigging is either done in water or air as a medium. Of the wet jigs the Harz or fixed-sieve type is the most commonly used both for ore and coal. Jigs in which the screen is moved up and down in a tank of water now have no application in the United States outside of the laboratory. Hand jigs employing this mode of operation are sometimes used for transient operations. In the Hancock jig the screen is movable and has a differential motion as to the horizontal component of the stroke as well as an up and down component. The horizontal component of the stroke moves the ore through the jig. This jig acts on the principles which have already been touched upon and on the factor of interstitial action the second principle under this section and described more fully under shaking tables at a later point. The effect of combining the two principles is to do away with the necessity of grading before jigging or the tendency within certain limits of size is to concentration according to specific gravity. The effect of mechanically advancing the material through the jig gives great capacity. At one place nine Hancock jigs displaced 132 Harz jigs occupying a floor space of 15,000 sq. ft. The nine Hancock jigs occupied 4,000 sq. ft. of space.

Air Jigs.—Jigs using air as a pulsing medium work under the disadvantage that they require close sizing to effect a concentration. If two substances of specific gravity s' and s'' fall through a fluid or gas with equal velocities then their diameters are in the proportion $D'/D'' = s'' - s/s' - s$ where s is the specific

[1] Consult Kirchoff's "Mathematical Physics" and Lamb's "Motion of fluids."

gravity of the gas or fluid. If s is zero (it will be negligible when representing the specific gravity of air) then D'/D'' equals s''/s'. That is, the free settling ratio cannot be greater than s''/s'. The free settling ratio of galena on quartz in water is 4.01. In air it becomes 2.08. Since bodies fall faster in air, a much greater number of pulsions can be given in an air jig, and by this means, the rapidity of separation and capacity can be increased to a point where theoretically it compares favorably with jigging in water. The principal factor which militates against the employment of air jigs is the cost of compressing sufficient air to give capacities which will compare favorably with wet jigs. With the possible exception of the Plumb air jig which attained some prominence some years past the mechanical details of such air jigs which have been invented have been unsatisfactory. The advance of the ore entirely by gravity tends to reduce capacity. The great air pressure which would be required to lift other than sand sizes limits the air jig to these sizes. The dust which would be made in the fine grinding that this implies cannot at the present stage of the concentrating art be satisfactorily treated by dry methods so air separation methods as primary methods cannot usually be given much consideration in devising concentration schemes.

Harz Jigs and Jigging.—In discussing the screen and jig ratio previously in this section hindered settling ratios were given as governing the range of permissible screen size which should be fed to a jig. It must be plain that since these ratios apply to pure minerals much closer sizing must be employed if there is much disseminated material in the gangue or in other words there must be close sizing if a middling product is to be drawn from the jig as well as a tailing. If concentrates are made on the jig and no clean tailing can be made the question of close sizing is not then of very great importance. Of late years the tendency is away from close sizing and indeed away from jigging altogether. This trend is partly due to the cost for the screening equipment, its relatively large upkeep and the labor of attendance for it and jigs. Very skilled and conscientious labor is required for jig operation. None of these adverse factors would weigh if good screening obtained in the mills and the jig would effect the close separation that is desired. Good screening can be obtained in the mills but it is not often found. The reason it is not found is because the screens are too heavily loaded.

In the matter of the jig not removing all the middlings the fault is not in the principle employed for as will be shown a little later it is capable of effecting separations where there are only minute differences in specific gravity, but in the mechanical details of the jig. For coarse feeds the only practical means which have been devised for removal of concentrates and middlings is the top discharge in some form.

The only way these discharges can be operated successfully is to let the material in any compartment collect until it is very much richer than the material entering it. The material removed in any compartment must have greater average specific gravity than the material entering it. This mode of operation does very well so long as there is heavy material to be removed such as galena, blende, etc. but when the light middlings containing only small proportions of such substances are to be taken out the top discharge method fails. The top discharge consists of a pen which dips down into the bed and is supposed to exclude in withdrawal all except the lower rich layers. The exit from the pen is at a lower level than the surface in the beds outside the pen to provide for differences in specific gravity inside and outside the pen and thus to provide a velocity head for removing the rich lower layers. In the concentrate removal

compartments the superior specific gravity of the material being removed will prevent any low grade material from the top layers in the vicinity of the pen from working their way down and underneath its lower edge. The discharge will draw from all parts of the "bed" or lower layers. In the middling compartments the slight differences of specific gravity between the top layers of tailing grade and the lower ones which it is desired to draw off is not sufficient to prevent the top pieces near the pen from being drawn down by the pull of the discharge. The discharging force is so feeble that it will pull from the top layers in the immediate vicinity of the pen rather than from points distant in the bed from the pen and there is not enough settling ratio left to overcome this tendency.

Tailings will go into the middlings and since they displace middlings the latter will go to waste. For difficult middlings-tailings separations a multi-compartment jig is required.

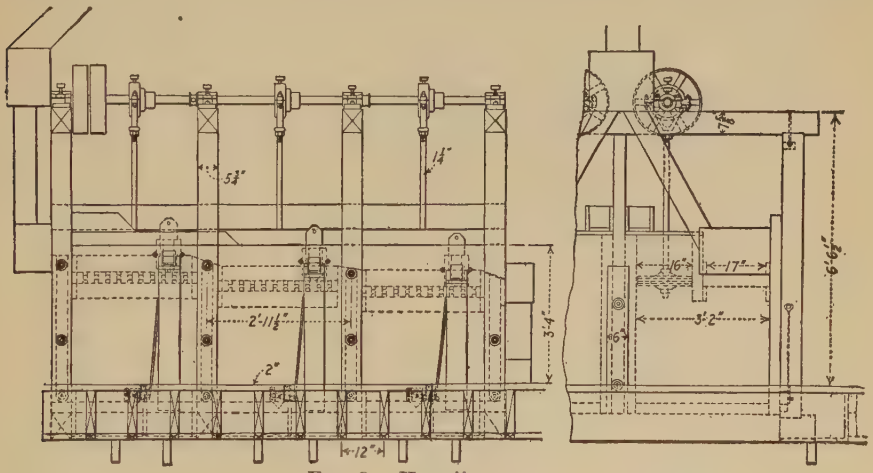

FIG. 3.—Harz jig.

That the Harz jig is capable of making very close separations is proved by the following experiments by the writer on material which contained but little over 0.50 per cent metallics consisting of lead and zinc and but a trifle in the way of silver reckoned as percentage. This material was jigged not for its base metal content but in hopes of finding a way of reducing the silver content. The ore consisted of gray copper, galena and zinc blende. Much of the silver was intermixed with or closely associated with the galena but there was also isolated amounts of gray copper. The ore was very clayey. In the tests material ranging in size from 1 to ¼ in. was fed to the jig. The large range of size fed at one time makes the tests all the more remarkable. In preparing the material for the tests it was screened dry and there was much material in the form of clay balls which disintegrated in the jig and flowed away from the tanks which were assembled around the jig for catching the products made. This slime which went thus to waste will account for most of the losses which are summed up under the head of "unaccounted" in the tabulation which follows. In preliminary tests an endeavor was made to run the top discharges of the jig but the material issuing was found to differ very little from the feed in point of assay. In the test which is tabulated the test jig was fed at the rate of 1 ton an hour for periods of 20 min. The beds were then removed and the beds skimmed. The beds were then thrown back and the test was resumed. Four sub tests were so made and the summary is the result of weighing and assaying the beds so removed, the hutches or material which

passed through the compartment screens while the sub tests were being made and the tailings made during them. The bed top is the material remaining on the jig screens after the last bed was skimmed following the completion of the last sub test.

Product	Dry weight pounds	Per cent weight	Assay			Recoveries, percentages		
			Silver, ounces	Lead, per cent	Zinc, per cent	Silver	Lead	Zinc,
Feed	2,453½	100.00	3.14	0.188	0.250			
All beds	393¾	16.05	7.36	0.644	0.798	37.16	56.74	51.22
Last bed top	98½	4.01	1.64	0.06	0.23	2.10	1.30	3.75
Hutches	95½	3.89	2.40	0.43	0.45	2.97	8.91	7.01
Tailings	1,644	67.01	1.86	0.078	0.137	39.48	27.83	36.70
Unaccounted	221¾	9.04	no assays			17.84	5.22	1.31
Totals	2,453½	100.00				100.00	100.00	100.00

The assay of the different beds of the subtests as to silver were:

	Test 40, silver, ounces	Test 41, silver, ounces	Test 42, silver, ounces	Test 44, silver, ounces
Bed compartment 1	17.60	18.60	14.40	8.60
Bed compartment 2	8.72	4.00	7.60	5.00
Bed compartment 3	6.24	4.00	5.80	5.92
Bed compartment 4	4.28	3.96	4.68	3.60
Bed compartment 5	6.96	3.08	5.44	5.60

The net area of the screen compartments was 11 by 16 in. The depths of the beds were respectively from compartment one to five, 6 in., 5 in., 4½ in., 4⅛ in., 4½ in. The respective strokes of the plungers were 1½ in., 1 7/16 in., 1⅜ in., 1 5/16 in., 1¼ in., the number of strokes 175 per minute. Jig practice where the lead content in tailings is reduced to 0.5 to 1.5 per cent lead is considered good.

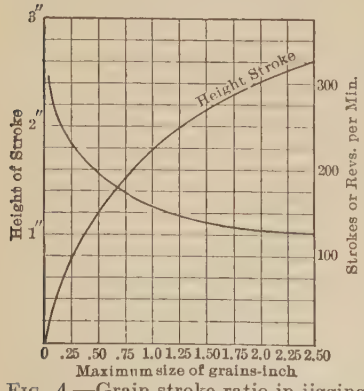

FIG. 4.—Grain-stroke ratio in jigging.

Working Data for Harz Jigs.—A Harz jig of the type commonly used in Western ore mill is shown in Fig. 3. The proper length of stroke and the proper number of strokes for various sized grains is shown by the curve Fig. 4. For extra close work and in lack of other data the capacity of a jig in tons per 24 hours per square inch of screen surface may be taken as low as $(d)^{\frac{1}{2}}/100$, where d is the average diameter of grain fed in millimeters. For roughing work and on easy problems the capacity figure given by this formula may be greatly exceeded. It may run to as high as $(d)^{\frac{1}{2}}/20$. The average capacity is nearer to the last figure.

Sand Tables and Concentrators.—Employing principle (2) given at the beginning of this section, the Wilfley table was the modern table of this type and is still one of the leading makes. Figure 5 shows the general arrangement of this table. The surface of the table is covered with linoleum to which riffles are secured. Feed enters at the point *A* and the separated material fans out, the leanest material leaving the deck at a point somewhere in the neighborhood of point *B* and the richest in the neighborhood of point *C*. The ideal arrangement of the discharging material is illustrated diagrammatically by Fig. 6. The black circles are the heavy mineral and the light the gangue. In order to have the clue to the arrangement of the grains, stratification must be explained and defined. This term is a misnomer, for there is no actual arrangement of distinct layers. Actually after the ore and water has reached the deck and come under the advancing shake of the head motion it tends to assume some such arrangement

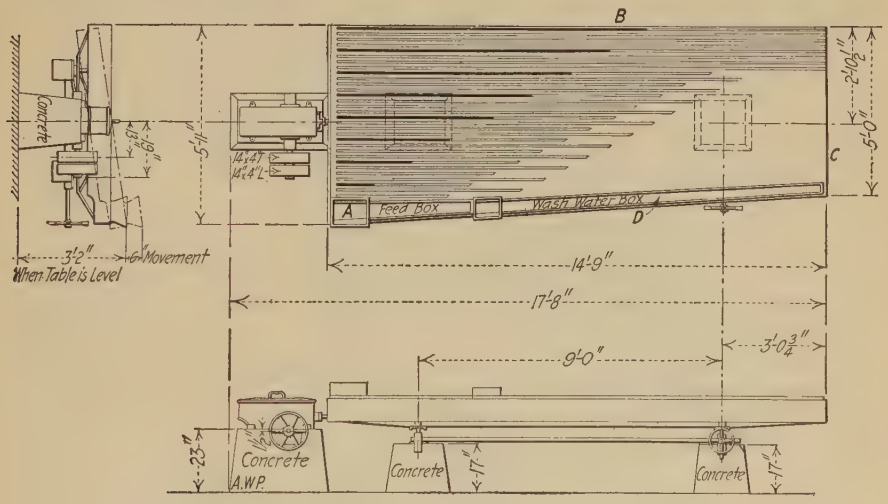

FIG. 5.—Wilfley table.

as is shown by Fig. 7. The riffles on the deck have little effect in causing this arrangement. They aid in holding the water and keeping the mass mobile and of even depth. This arrangement of the grains is caused by interstitial settlement, which has already been explained to some degree in Section VI. The additional effect of specific gravity is to tend to make the heaviest grains pass down and take a position nearer the deck than those of inferior specific gravity. When the ore is spread out sidewise by the inclination of the deck and the influence of the wash water which comes from the water box *D*, Fig. 5, it tends to assume, where there are two minerals involved in the concentration, the appearance shown in Fig. 6. The part of this figure enclosed in dotted lines should be compared with Fig. 7.

The capacity of shaking tables ranges from 5 to 20 tons per 24 hr. One man can attend 20 of them. Owing to the overlap of the bands in making separations on sand tables there cannot be a clean division of the products. Overlapping is greater where a screen feed is placed on the table and less where a water classified feed is employed since this tends to produce small grains of heavy mineral and large grains of coarse ones. Expressed diagramatically it would tend to bring heavy grains such as *E*, *F*,

and G onto the deck along with grains of the lighter kind such as M, L, K, J and I, avoiding the overlap.

For separations below 80-mesh size and for slime separation and particularly for pilot-table work following flotation the Deister slimer table shown in Fig. 8 is very satisfactory. Owing to its extremely powerful differential motion it requires very rigid foundations, concrete being the preferable material for this purpose.

Canvas tables stationary and revolving and ordinary revolving round tables and buddles employ the third principle of concentration listed at the beginning of this section. In modern metal-milling practice these devices are falling into disuse with sulphide ores owing to the advance in flotation practice. The principle effects a separation by a film current flowing down an inclined plane or curved surface. Since the upper layers of such a film are more actively moving than

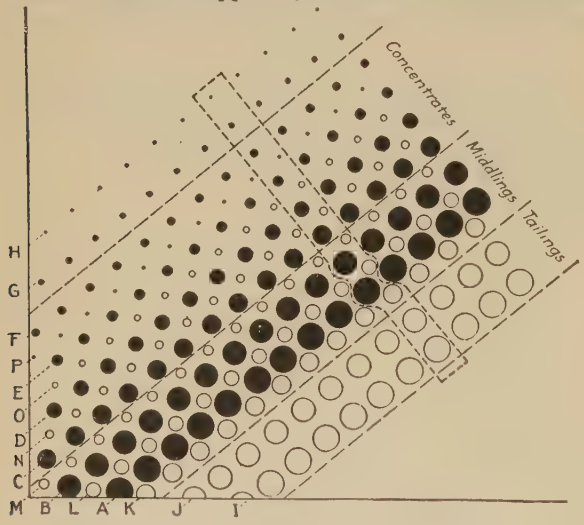

Fig. 6.—Diagrammatic view, discharge corner of Wilfley table.

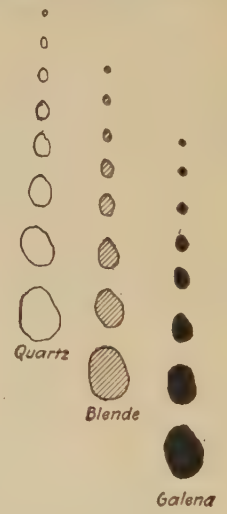

Fig. 7.—Hindered settling of unsized material.

the lower large grains are more readily affected by it and transported by it than small. The greater also the specific gravity of the grain the less other things being equal it will be affected by the current. A small grain of high specific gravity will cling best to the separating surface. If classified feed receives treatment on this sort of device it would be expected since the gangue grains are larger that these would be carried first to waste, the grains of greater specific gravity moving more slowly or standing fast. On canvas stationary tables, the slime feed pours down the separating surface for a certain period of time after which it is diverted onto an unused deck and the remaining sand which contaminates the concentrate is washed down with a limited amount of wash water. Following this wash the concentrate is removed by washing down with strong jets. For very slimy substances there should be about 2 lb. of water to the square foot of cleaning surface and a slope of 1¼ to 1½ in. to the foot. For coarser material less slope should be employed and 10 to 15 lb. of water.

Magnetic Separation.— Magnetic separation dates back to 1847 or earlier and up to the advent of the Wetherill machine, the use to which the method was

put was for separating highly magnetic substances such as magnetite and magnetic pyrites. Wetherill's patent (U. S. 555792 March 3, 1896) was the first one to show mechanical means for producing a concentrated magnetic field, so that such feebly magnetic material as spathic iron and the ferruginous forms of blende could be attracted to the pole pieces. Magnetic separators either act on

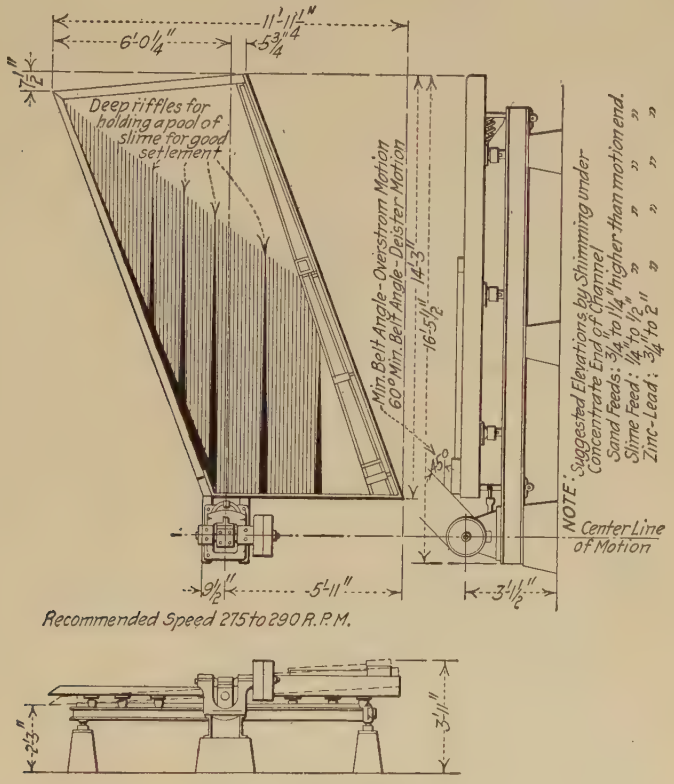
11'-1 1/4"
6'-0 1/4"
5 3/4"
1 1/2"
Deep riffles for holding a pool of slime for good settlement
Min. Belt Angle-Overstream Motion
60° Min. Belt Angle-Deister Motion
16'-5 1/2"
14'-3"
NOTE: Suggested Elevations by Shimming under Concentrate End of Channel
Sand Feeds: 3/4 to 1 1/4" higher than motion end.
Slime Feed: 1/4" to 1 1/2" " " " "
Zinc-Lead: 3/4" to 2" " " " "
Center Line of Motion
9 1/2"
5'-11"
3'-1 1/2"
Recommended Speed 275 to 290 R.P.M.
2 3/4"
3'-11"

FIG. 8.—Deister slimer table.

dry material or wet. Designs for treating dry material are the far more common. The principal high intensity machines are the Wetherill and the International. Magnets may be either fixed or movable. The Wetherill and the International belong to the first class. The dropping of the magnetic material is effected by causing it to pass outside of the magnetic field. This is done by: (1) preventing the magnetic material from coming in direct contact with the magnet by means of a traveling belt, or a revolving cylinder of nonmagnetic substance, to which the magnetic material will cling so long as in the magnetic field, but from which it will drop as soon as removed from it. (2) The magnetic material may be attracted directly to the pole, which by revolution or change in electrical connection may suffer a change in polarity, or become non-magnetic, thus dropping the attracted particles after removing them from the stream of non-magnetic. (3) The magnetic material may be attracted directly to the pole, and be removed therefrom by means of a brush or scraper.

The chief feature of the Wetherill machine consists of two magnetic poles placed in very close contact with one another. A main conveyor belt carrying the material to be separated passes between the poles. Cross belts at right angles to the main belt also pass between the poles for carrying off the magnetic material. These are in contact with the upper pole and the magnetic material clings to the lower side until without the influence of the magnet. The upper pole is bevelled so as to concentrate the lines of force and is the effective attractive pole. In zinc-cleaning problems the Wetherill may require as much as four amperes of current, with 30,000 ampere-turns. The capacity of the Wetherill ranges from 700 to 1000 lb. per hour and the machine will consume 7.5 hp.

In the International separator a cylindrical armature made up of thin laminated discs of a special annealed wrought iron and mounted on a steel shaft revolves horizontally between the pole pieces of a large inverted horseshoe field magnet. By induction the magnetism of the pole pieces causes magnetic poles to appear on either side of the surface of the armature. Material to be separated is fed from the hopper upon the top of the revolving armature and is carried by its movement around the armature. At the bottom position where the polarity changes the last of the material clinging to the armature will fall off as there is no attraction there whatever. The magnetic particles will fall off through an arc of 90 deg. in the order of their attractability and by suitably arranged hoppers below separations can be made taking advantage of attractive differences. In the Wetherill machine three sets of poles effect the same purpose. The Wetherill machine in the test size is very convenient for testing as it can be readily adjusted for giving high or low intensities.

The range of magnetic separators is about the same as shaking tables, from 12 to 80 mesh. They fail on dust or very fine material. The Western ore mills they are most commonly used for effecting the separating the mixed pyrites-blende product which comes from shaking tables. This owing to the slight specific gravity differences always comes off the shaking tables badly intermixed. If the blende be ferruginous it may be possible to separate it from the pyrities. Usually however it is necessary to roast it to magnetic pyrites after which a low intensity machine of the Dings or other type may be used. It is very difficult to roast pyrites and get the resulting product all into the magnetic iron form. Roasting and treatment of table products requires dryers, roaster, coolers, magnetic separators and the bins and necessary accessory apparatus. In many cases the equipment and operation expense does not make the separation game worth the candle.

Electrostatic Separation.—Electrostatic separation is due to the work of Blake and Morscher. The improved Huff machine has an all metal frame and static electricity is provided by a 4 hp. motor-generator set, the generator being rated at 300 volts and 5 amp. The transformer used delivers an alternating current to a revolving rectifier or interrupter mounted on the shaft of the generator. By this rectifier either all the crests of the alternating current wave, which are of course, of like sign or all the troughs could be used by the separator if the rotation of the apparatus is reversed. The voltage at the separator is between 20,000 and 30,000. The actual separating unit of the device is a statically charged round metal bar of small diameter. This can be used stationary or revolving if a wide spread of repelled material is desired. In either case the material to be separated is delivered at its top and slides down over the curved face. Opaque substances with a metallic luster are repelled and earthy materials are attracted. Some of the more common minerals which are repelled (conductors) are native metals, pyrite, chalcopyrite, galena and other sulphides. Even hornblende since it has a lustrous surface and some metallic content is repelled though feebly. The non-conductors are the earthy materials and gangues and such substances as pure zinc sulphide or resin jack which has no metallic luster. Resin jack has been made a conductor by treating it with a weak solution of copper sulphate.

The chief commercial application of the method is in separating blende with little or no occluded pyrite, from the latter. The process avoids the roast necessary for separating these substances magnetically. The process cannot be applied to material coarser than six mesh when the capacity of the standard Huff machine of about 10 ft. height and about 5 ft. width may be as high as 35 tons per day of 24 hr. The rougher portion of the separator consists of six separator bars which splits the entering stream into a conductor and non-conductor portion. The finisher part of the unit consist of two parts of six bars each, one set being for conductor material and the other for the non-conductor material. The middlings made by each side are returned to the top of the finisher for retreatment. The machine will not treat the finest dust and on material ranging from 80 to 200 mesh the capacity under favorable conditions may be as high as 4 tons a day.

Flotation.—Flotation takes advantage of the affinity of sulphide mineral and other substances for oils, greases and gases or of the repulsive effect of the same substances on the surface tension "skin" of a water surface. Modern applications of the principles employed in flotation fall under three heads. (1) The direct floating of powdered dry metallic substances on the surface of water. Some success has attended the use of this method with graphite and molybdenite. (2) Recovery of valuable material by a grease-coated surface. This method has been successfully used for recovering diamonds in the South African field. Haultain and Stovel employ a greased endless belt. At a trial of their invention at the Mammoth mill in the Coeur d'Alene region, Idaho, the endless belt with a cotton center and wool edges was placed in a highly inclined position and fed by a down-pouring stream of slime the belt moving upwards against the flow. Mineral coated grease was taken off at the top of the belt and removed to a steam heated tank filled with water where the grease was melted and the concentrate settled to a removing worm at the bottom. The grease after being floated off the water was congealed and again applied at the bottom of the belt. (3) The methods most commonly employed consist in making an air-oil-sulphide froth which rises to the top of the water by its superior buoyancy where it can be skimmed off or caused to flow to any desired collection point.

In the Callow process air is introduced from a compression source through a porous diaphragm in the bottom of the series of tanks or compartments employed. The air bubbles in their ascension become oiled by the oil added to the stream of slime flowing into the apparatus and by the superior affinities of the air, oil and sulphides for one another form a "froth" so that a substance like beaten white of egg rises to the surface of the separating compartments through its superior buoyancy The oil and sulphides form the "framework" of the froth Some gangue is also carried up with the froth but the amount of this compared to what goes to waste from the bottom of the apparatus is small. In the Ruth apparatus there are separate aëration chambers and the aëration means consist of a hollow vertical shaft driven by belting or other means. Air is pulled down through the hollow shaft by means of the centrifugal force created by a disc secured to the bottom of the shaft and with ports at the periphery connected with the central opening in the shaft. The slimes and oil are introduced into the aërating compartment and pass from this into a compartment facing it where the froth rises and discharges.

In the chief class of flotation machines the air is actually beaten into the oiled froth by revolving beater or impellers. Beaters mounted on horizontal shafts have received much attention from inventors in recent years. They revolve at lower speeds than the machines using vertical impellers and are arranged to trap the air by having the

beaters come above the surface of the water. Although these machines run more slowly than impeller types with vertical shaft the wear and tear on the beaters is more severe than on the impeller type. In this type the impeller not only serves for beating in air but it has the further function of pulling the pulp, except the portion which rises as froth, from the spitzkasten or compartments facing the impeller compartments provided for froth rising, to the impeller compartment. In the multi-compartment machines employing the Minerals Separation process the pulp passes into the first impeller compartment and from this it flows into the first spitzkasten where the first froth rises. The general level of the impeller compartments is the same throughout as is also that of the spitzkasten but that of the latter is lower than the impeller compartments. The material which does not float in the first spitzkasten is drawn up into the second impeller compartment by the pull of its impeller. It then flows into the second compartment where second "floats" are obtained the balance being pulled to the third impeller compartment, etc.

This allows of varying the treatment in various compartments, routing of "floats" of the different grades from the different compartments to different points for separate disposal, advantages which are not possessed by the horizontal beater machines unless they be used in series with consequent loss of head room and multiplication of apparatus. The horizontal machine has the advantage that it takes in the air better and it is no difficult matter to secure any desired regulation. The vertical impeller type with high velocity of rotation or Minerals Separation type makes a much stronger elastic froth than any other type of machine and on high ratio of concentration problems this is a very desirable feature, particularly if the minerals to be separated are of high specific gravity. A strong froth is not so desirable a feature where there is present in the ore a large proportion of sulphides of relatively low specific gravity as the lack of air in many cases will make for low capacity. The tendency of the Minerals Separation Co.'s engineers has been to make the arrangements of the impeller compartment such that the pulp in it is as shallow as possible. This increases the entrapping of the air but affords no means of regulating it.

The general arrangements for a 12-cell Minerals Separation type of machine are shown in Fig. 9. A peripheral speed of 1,200 ft. per minute or thereabouts is best for the impellers. Four of these machines used as roughers handled up to 500 tons per day of 24 hr. This is a machine of small-size cell and was used on a high ratio of concentration problem. On a low ratio problem this spread of surface would probably not take care of more than 300 tons per day and fewer units and larger cells would be preferable. The froth material from the roughers went to two eight-cell machines of identical design. The waste from the finishers went to one 12-cell machine and the waste from this was returned to the 4 roughers for retreatment. The flotation concentrate passed to two pilot tables and the crystalline concentrates made were routed to two settling tanks from which they alternately drawn daily and shipped. The colloidal concentrate was pumped into a 30-ft. Dorr thickener and finished on a Portland filter.

Other Working Data on Minerals Separation Machines.—If 12-in. impellers are employed the power required to drive one impeller is about 1.75 hp. Larger size impellers may be reckoned as being proportional to the diameter, using the horsepower figure given as a base. The consumption in power with other types of machine is less than this.

Oils.—These are of two kinds the "collectors" which have high affinities for sulphides and the "frothers." Hard-wood creosote is the standard collective oil but owing to its expense other wood-, coal-, and oil-distillate products are substituted for it either entirely or in part, coal tar being a favorite. On silver minerals

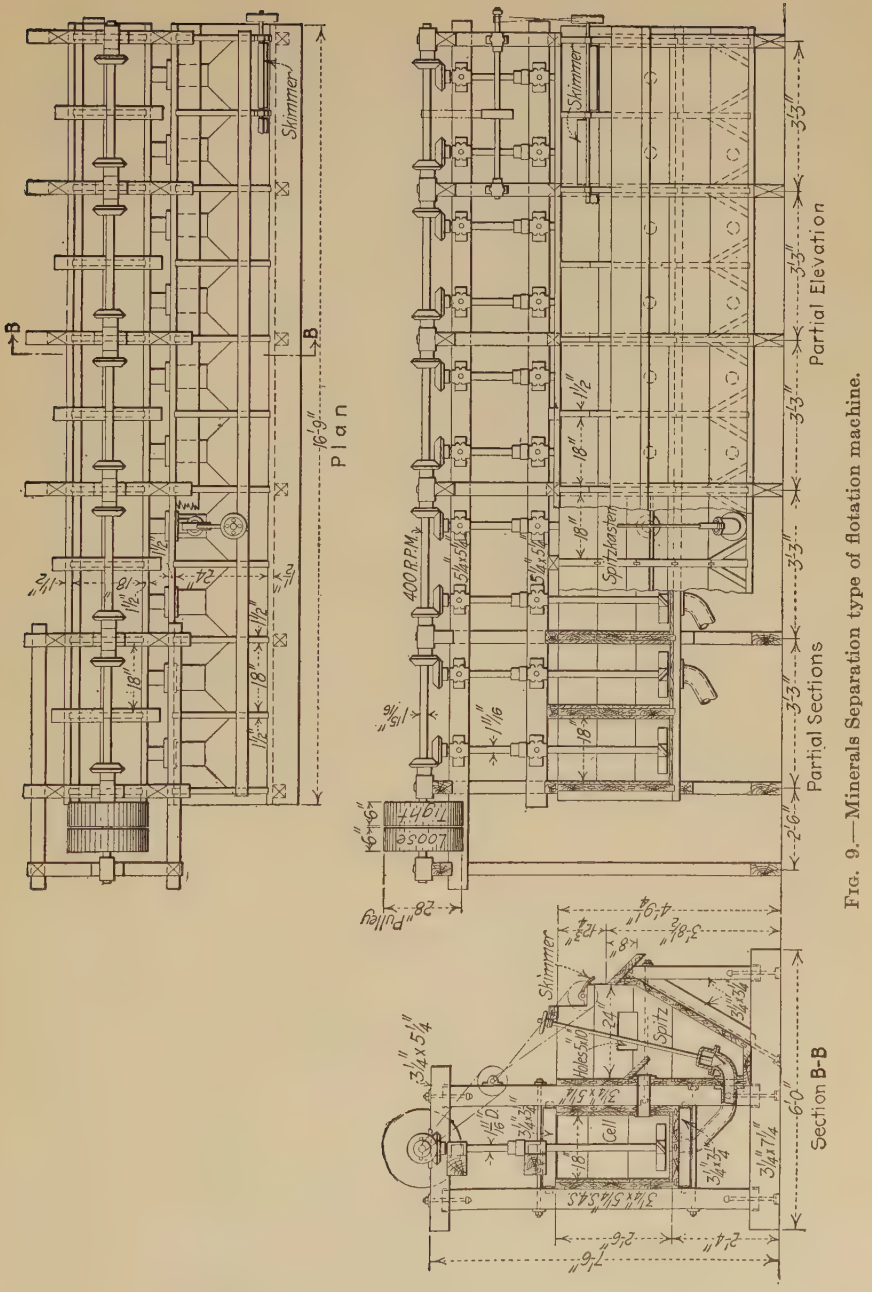

FIG. 9.—Minerals Separation type of flotation machine.

the use of a certain proportion of creosote can scarcely be avoided. The standard frothing oil is steam-distilled pine oil. Destructively distilled pine oil or other light volatile and miscible oil can often be substituted for it. The steam oil is very expensive. The frothing oils have an affinity for air rather than for the sulphides. They form the principal part of the framework of the froth though they form only a small proportion of the oil mix. Froth flotation made the flotation art a practical success as it was accompanied by the use of small quantities of oils. The average oil use is today below one per cent of the weight of ore treated and a half per cent use would be nearer the average use than the higher figure.

In the matter of preferential treatment the flotation art has advanced very little. The oils make very little selective difference in the sulphides floated and various attempts to intensify the selective effect have not resulted in much advance. As the art stands today it is increasingly ineffective on material which is coarser than 40 mesh largely because of occluded gangue matter which weakens the flotation effect. The recovery on the very finest slime is also unsatisfactory. Any sulphide ore which has not been exposed to oxidation will float well on being ground to 40 mesh or finer and on those portions of it which will settle promptly in water.

When the process was at its high tide of enthusiasm some years ago it was predicted that it would displace all forms of sand and slime concentrating apparatus. It was even predicted that it would become the only process which need be employed even on those ores from which concentrate could be obtained after a preliminary crushing, for the process was so simple, the extraction so high, etc., that it would overcome the costs and disadvantages of fine grinding. The truth of the matter is that the flotation process is largely but an adjunct to older methods of concentration and that is but an extra string to the bow of the metallurgical engineer. If results cannot be obtained from one mode of treatment an entirely different one will often yield results. If one concentration method will not yield a satisfactory modicum of concentrate and entirely different one may yield an additional one worth while. The old guiding law of concentration to "begin it as coarse as one can" is just as true today as it ever was.

SECTION IX

LEACHING AND DISSOLVING

By J. V. N. Dorr[1] and F. F. Peters[2]

The term "leaching" as used hereafter is taken to mean the dissolving of a constituent of a comminuted material and the separation of the resultant solution from the undissolved portion or residue. Two general methods are in use: first, percolation, in which dissolution and the separation of the dissolved material are done coincidently, and second, dissolution by agitation of a finely divided pulp held in suspension and separation of the solids by decantation or filtration or both.

In the latter case the dissolving is usually distinct from the separation of the dissolved material and the term "leaching" is often applied to the former. The theories of leaching especially those relating to dissolution will be first discussed, followed by their practical applications.

Theoretical Discussion.—The problem involved is almost identical with that of dissolving any solid in a suitable liquid and is governed by the same physical and chemical laws, if that chemical action which dissolves the solid is the only reaction taking place in the solution at the time. Dissolving salt in water is governed by the same laws which govern the dissolving of calcium carbonate by hydrochloric acid or the leaching or gold and silver from their ores by cyanide (see Ostwald on "Heterogenous Reactions").

The velocity of dissolution is dependent upon the following factors:

1. Physical condition or nature of the matter to be dissolved.
2. Velocity of diffusion.
3. The concentration of the solvent or in other words the strength of the solution.
4. The rate of change of relative position between particles of solid and the adjacent liquid.
5. Temperature.

Effect of Physical Condition.—The increased exposure of surface due to fine grinding will be discussed later but two interesting phenomena should be mentioned here. (1) When the finest portion of a suitable crushed soluble material is added to a saturated solution of the same it will often dissolve to cause supersaturation which effect could never be gotten by contact with the coarser particles. (2) The rate of dissolution for different faces of the same crystal usually varies. These facts point to the necessity of making tests on the actual material to be treated when considering a leaching problem.

In the theoretical discussion that follows it will be assumed that where the soluble material is contained in an insoluble matrix it is so finely pulverized that as dissolution proceeds all particles of the former will be exposed to the action of the solvent and our problem is to consider means of accelerating dissolution.

Effect of Diffusion.—The process of dissolution seems to take place in two steps; *viz.*, first there is a reaction between the solid and the solvent in immediate contact which is followed by the second step, the diffusion of the products of this

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[2] Westport, Conn.

action from the surface of the solid. It is assumed, that we have at all times in direct contact with the solid particles during the process of dissolution a thin film of saturated solution, and that the unsaturated solution never has contact with the solid but becomes concentrated by diffusion from this film.

It then becomes evident that the actual velocity of dissolution may be quite different from the observed velocity, due to the rate of this diffusion. There is no need to consider these two velocities separately except to explain some exceptional phenomena discussed later under the laws of mass action.

The velocity of diffusion is dependent upon the osmotic pressure exerted by the material in solution. If some copper-sulphate crystals are placed in the bottom of a jar of water they will dissolve and a layer of dark-blue solution will form in the bottom around the crystals which will diffuse up through the rest of the solution very slowly until it is all of the same concentration. It required a perfectly definite force to lift this heavier material up through this solution and this force was osmotic pressure. Therefore, more highly ionized solutes should diffuse through the solvent more rapidly than slightly ionized materials as the osmotic pressure exerted by the latter is less than the former. The second, third and fourth factors listed above as affecting the speed of the dissolution are related together by the law of mass action following.

Law of Mass Action.—Since dissolution of a solid in liquid is a chemical reaction it must be governed by the simple laws of mass action and therefore it is possible to develop an expression concerning the velocity of this reaction. Ostwald, Brunner and others have done a certain amount of work on this subject and the following discussion is derived largely from their work.

The speed of dissolution is directly dependent upon the amount of surface exposed to the action of the solvent and upon the concentration of the solute in the solvent. Therefore, if we let

S = surface,

x = the concentration of the solution at any time,

a = the concentration of the film of saturated solution in direct contact with the solid,

t = time, dx = very small change of x,

K = constant—dependent upon solvent, dt = very small interval of time,

we may obtain the following expression:

$$\frac{dx}{dt} = KS(a - x) \tag{1}$$

By integrating this expression we obtain

$$1/t \log_e a/a - x = KS \tag{2}$$

This formula will only hold true if S remains constant. But it has been found in many cases that though the surface exposed did change materially, still the rate of dissolution KS remained constant. This would seem therefore to show that the second step (velocity of diffusion) here were controlling the action. In other words the diffusion of the solid from the saturated film to the solution itself was so slow that the surface exposed was more than enough to resaturate the film with solid as it diffused to the rest of the solution, and this fact supports the assumption made earlier in the paper that the process of dissolution is made in these two steps.

Returning to the formula above (No. 2) it will be seen that the value of x (that is, the concentration of the solid in the solution) varies with time according to the logarithmic curve, Fig. 1. We would naturally suppose that the speed of dissolution would decrease as the solvent becomes more saturated with the solute but it is of interest to note the curve that relates to these two factors.

In Fig. 2 are given two curves taken from some tests on leaching copper ore with ferric sulphate which corresponds to the theoretical curve of Fig. 1. Curves *A* and *B* were obtained by plotting time against percentage of extraction. As the tests

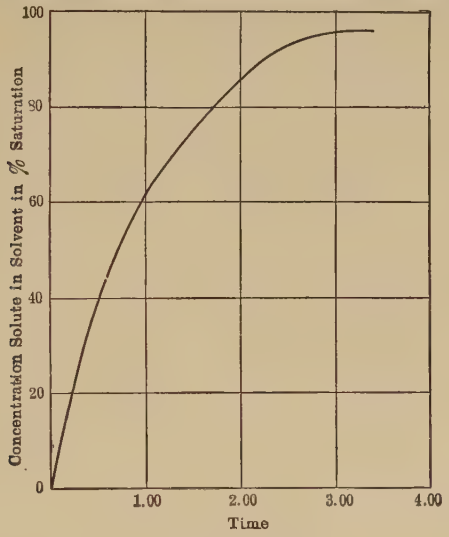

FIG. 1.—Rate of solution.

were made by intermittent agitation the percentage extraction here corresponds to the concentration of solution in Fig. 1, the concentration of the solution is directly dependent upon the amount of solids extracted. Curve *A* was made on one type of ore and *B* on another and yet the only difference in the curve is a vertical displacement due to the difference in the relative rates of dissolution.

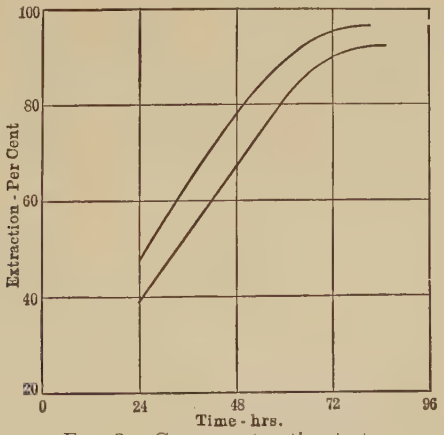

FIG. 2.—Copper extraction tests.

If we allow the process of dissolution to continue without setting up any artificial currents to aid diffusion this concentrated film in direct contact with the solid is going to become thicker and thicker and the speed of dissolution is going to become slower and slower as the thickness increases. This is not due to the effect of mass action alone but also because it takes a definite period of time for the dissolved particles to

pass through the surrounding unsaturated solution. Therefore to increase the speed of dissolution we must decrease the size of the film (and especially in cases where the speed of solution is chiefly dependent upon the speed of diffusion as in the case quoted above) by setting up a flow of solution past the solid particles. Few experiments have been made in this connection due to the difficulties encountered in getting

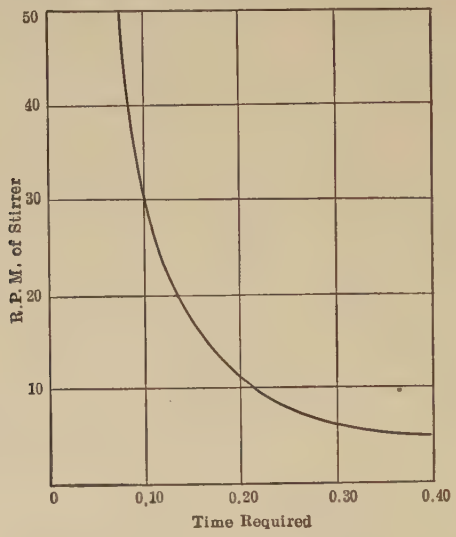

FIG. 3.—Effect of agitation on solution.

any exact data. However if L be the thickness of the concentrated film around the particles of soluble material we get from our first expression

$$dx/dt = KS\ (a - x)$$

or

$$1/t \log_e a/a - x = KS/L \qquad (3)$$

and since L will depend upon the number of revolutions N of the stirrer used to agitate the mixture we should by experiment be able to find a relation between L and N.

In an experiment on dissolving benzoic acid in water it has been found that

$$1/t \log_e a/a - x = BN^{3/5} \text{ where } B \text{ is a constant}$$

or as $\log_e \dfrac{a}{(a-x)}$
is a constant in any one case we find

$$T = \frac{c}{BN^{3/5}} \qquad (4)$$

By substituting different values of N we then get the curve shown in Fig. 1.[1]

The power consumed by an agitator varies with the speed of agitation and Fig. 3 is a curve derived from experimental work to determine the relation between dissolution and speed of agitation. From this we see consideration must be given to the cost of power as against agitation capacity as decreased time in agitation means greatly increased power consumption.

Agitation is necessary so long as there is any soluble solid present due to the fact that the film is continually being reformed. However, when all the soluble matter

[1] The values of time obtained here are valuable only in so far as they give us a graphical illustration of the effect of increasing the velocity of the solvent past the particles. A much more valuable experiment would be to suspend the particles by means of threads or wires in the cross-section of a tube and vary the flow of the solution so that there would be no question as to the efficiency of the stirrer in forcing the solution to move past the particles.

has gone into solution we have quite a different state. The film above referred to not being continually reformed disappears and the entire solution becomes homogeneous within the pores of the solid as well as without.

The above applies more particularly to agitation for dissolution where the soluble component is a large constituent of the material to be leached. In many cases as in the treatment of gold ores by the cyanide process it often happens that the volume of the metal to be recovered is an infinitesimal portion of the whole. For instance, in recovering $5 per ton or ¼-oz. of gold its volume would be about one-millionth and this would occur largely in minute fissures so that the time required for diffusion through the pores of the insoluble material will be the limiting feature and a violent agitation will be useless.

Temperature.—The general effect of increased temperature is to increase the velocity of dissolution irrespective of the increase when the substance is more soluble in this hot solvent than the cold. This increase is small and does not warrant heating solvents. For instance salt in water has a solubility curve which varies little with temperature and though the velocity of dissolution in hot water would be slightly greater than in cold, the difference is not found to be sufficient to increase the fuel consumption to heat the solvent (see table of Solubilities, Appendix).

Selective Dissolutions.—By selective dissolution is meant the process of dissolving one compound to the exclusion of another from a solid made up of two or more compounds classed as soluble. The following means are resorted to in this work.

1. Time of contact of solution with solids.
2. Temperature.
3. Solvent.
4. Use of permeable membranes.

In applying these it is generally a combination of one or more of these methods that is used.

In leaching tan bark for instance it is found that the first two, *i.e.*, time and temperature, affect the leaching. The tan bark contains in addition to the tannin which we wish to leach from the bark some coloring matter which is soluble and would color the leather if it got into the extracts. The coloring matter however is only very slowly soluble in water of about 170°F. though it is easily soluble in water of higher temperatures. It is necessary, however, to heat the solvent (water) to about 170°F. to extract the tannin. Experience has shown with proper temperatures that when the liquors begin to color up the tannin content should be about right, due to their relative speeds of dissolution. If the solution is too hot however, they will begin to color up before the tannin is up to strength as the velocity of dissolution of the coloring matter increases with temperature more rapidly than does the velocity of dissolution of the tannin.

Solubility tables are naturally utilized for regulating selective dissolutions but in using them the effects of other soluble compounds on the solubility of the compound in question must be determined as its effect may be quite marked. For instance in Fig. 4, Curve 1 shows the solubility of sodium sulphate in pure water while Curve 2 shows its solubility in a solution of salt.[1]

The third method (that is, the use of suitable selective solvents) is almost universally used in effecting selective dissolution. It may be used in connection with the temperature effect as in leaching *caliche* (nitre ores) or alone as in organic work.

[1]There are some published tables in Comey, Seidell and the "Tables Annuelles" of the International Union of Pure and Applied Chemistry giving solubilities of two and three salts simultaneously but in general experimentation is necessary in any problem involving complex solutions.

Caliche may have about 20 per cent sodium nitrate and 25 per cent sodium chloride. Now if we look at Fig. 5 we see that one method of treatment would be to leach the ore at a temperature of 80°C. At this point we would dissolve 118 parts of sodium nitrate and 13 parts of sodium chloride per 100 parts of water. Now by cooling this

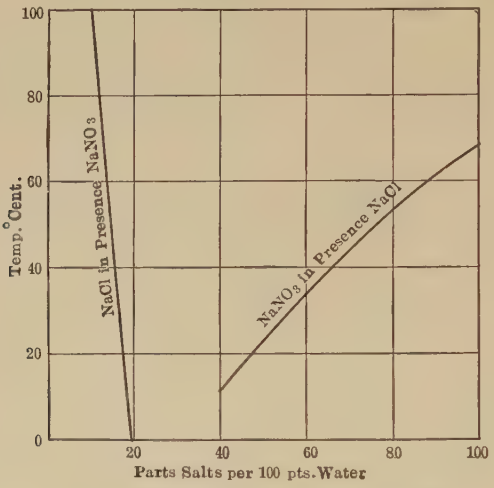

FIG. 4.—Effect of another salt on solubility.

solution to 20°C. the solution would hold 26 parts sodium nitrate or 118 − 26 = 92 parts, sodium nitrate will be precipitated and as sodium chloride is more soluble at 20°C. than at 80°C. no salt would be precipitated. Therefore theoretically we should get 100 per cent sodium nitrate. Due to the problem of getting water in Chile and

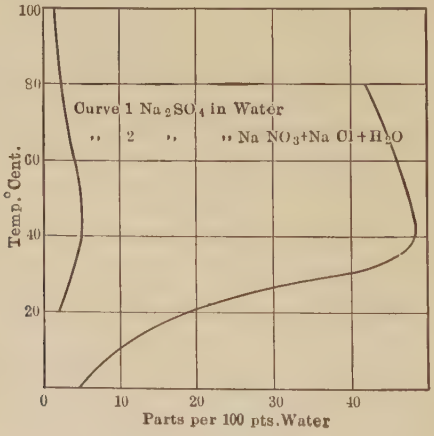

FIG. 5.—Sodium sulphate-sodium nitrate solubilities.

fuel expense this process is peculiarly useful as the salt and weak nitre solution may now be used for dissolving additional niter with no evaporation, except that required to concentrate wash liquors.

Permeable Membranes as Applied to Selective Dissolution.—In most of the leaching and dissolution processes the material is shredded or ground so as to

expose the soluble particles to the action of the solvent. In the leaching of sugar from beets however, just the opposite is sought. In the beet we have two types of solid particles, crystalloids and colloids. We wish to dissolve the crystalloids and prevent the colloids getting into our solution. Crystalloids will pass through permeable membranes whereas colloids will not, or only very, very slowly. Now the sugar beet is made up of small cells containing the sugar and other crystalloids in solution with the colloids and these cells have walls of a very fine membrane. Therefore if when slicing the beet these membranes are not damaged, when brought in contact with water the crystalloids will pass through the membranes but the colloidal matter will be held back thus keeping the sugar solution clear.

Another case of interest in the beet-sugar industry of the application of osmosis is in removing objectionable salts from molasses. It is found in sugar refineries that certain salts are taken into solution which tend to prevent crystallization as they become more concentrated. A molasses will be obtained therefore containing a very high percentage of sucrose but on evaporation the sugar does not crystallize due to the effect of these contaminating salts. These salts however are highly ionized, while sugar is not. They will therefore diffuse through a permeable membrane much more readily than sugar. Molasses containing these salts is placed in contact with permeable membrane on the other side of which is water. The salts then tend to flow through the membrane to the water and thus their concentration in the molasses is reduced, while the loss of sugar through the membrane is small.

Continuous Agitation.—The art of conducting leaching operations involving agitation continuously, has been well known and practiced in various metallurgical operations for quite a number of years, but has only recently been introduced into the chemical field. To design a continuous agitation system, we must first know the amount of material to be treated in any unit of time and secondly, the time of contact necessary to effect complete dissolution. We would then provide sufficient volume in our agitators to retain the material for this period and would feed and withdraw material at an equal and uniform rate.

A continuous agitation system may be made up of one or a series of agitators, depending on the time of retention necessary. A great deal has been written on the theory of continuous agitation, and especially on the possibility of untreated material being discharged prior to complete extraction. Of course, it is apparent that the material is not retained intact for the computed period, but that this represents merely an average time of detention. In fact, a portion of the feed at any instant will be discharged almost immediately and on the other hand, from the purely theoretical standpoint, another portion will remain in the system an infinitely long time.

In actual practice continuous agitation is seldom attempted in a single tank, but a series of three, four or even more, smaller agitators used in order to minimize the "short circuiting" effect. The number of tanks necessary depends primarily on the period of detention necessary. Several articles have been published in which an attempt has been made to derive a mathematical relation between the time of contact, number of agitators necessary and percentage of extraction.

These articles lead one to believe that continuous agitation can only be effectively applied by employing a prohibitive number of agitators of large capacity and yet the actual results obtained during the past 10 years in hundreds of metallurgical and chemical plants do not corroborate this.

We believe the discrepancy is due to the following points which are not taken into account in these various formulas. First, if we assume we have a continuous agitation system made up of four agitators, in which we are leaching pyrites cinders with 10 per cent sulphuric acid the system will become balanced and the strength of the sulphuric acid in the four tanks will become constant. For purposes of comparison, we will assume this to be:

First, tank 9 per cent H_2SO_4; second, tank 5 per cent H_2SO_4; third, tank 3 per cent H_2SO_4; fourth, tank ½ per cent H_2SO_4.

Now, assuming a detention period of 24 hr. is necessary the average period in each tank would be 6 hr. Also the rate of dissolution as shown by Fig. 1, is not a straight line curve. (Figure 1 is plotted against time but in intermittent tests of this kind the solution is becoming weaker and is the real variable while time is only the apparent variable.) The first tanks of this system would then be working on the steep part of this curve while the last tank only would be on the flat portion.

Now, as pointed out before, 6 hr. is the average time of detention in each tank but some material will be retained in the first tank as long as 12 hr., but the rate of extraction throughout this period will be greater than if it were carried on by intermittent agitation due to the fact that the strength of the solution remains constant. This rate may be gotten by drawing a tangent to the curve at the point corresponding to the concentration of this first tank.

Second, one other point not brought out in these calculations is that we seldom obtain a 100 per cent extraction in commercial operations and usually aim at a point in the curve before it flattens out. If our leaching experiments gave us a curve similar to Fig. 1, we might aim for a 94 per cent extraction. A 100 per cent extraction, of course, is possible, but requires a longer contact than we would ordinarily be willing to provide. However, in continuous agitation we will obtain a higher extraction on some material which will offset a part of the loss due to the low extraction on material leaving the system prematurely.

Finally, to sum up these two effects, we believe the "short circuiting" of material in continuous agitation systems is largely offset by a portion of the material being dissolved at a rate higher than that obtained by intermittent systems and to a higher extraction obtained on material remaining in the system longer than the average period.

By counter current dissolution is meant a process of dissolving one substance in another by bringing them in contact either intermittently or continuously in such a way that their respective content of the soluble compound varies in a direct proportion. A specific case would be the leaching of beets in the manufacture of beet sugar. A battery of 12 or more large cylindrical tanks is used. They are connected in series so that water fed into the last tank flows through the other 11. When all the sugar has been extracted from the beets in the last tank this tank is cut out of the circuit and a fresh tank added at the other end. In this way the beets containing the least sugar come in contact with solution in a like condition.

This system is used in practically every leaching operation practiced on a large scale and effects very material savings in the amount of liquor used and solute lost over completed leaching of each tank. This system may readily be applied to all the ordinary methods of leaching and washing solids such as decantation; filtration; or lixiviation, and the deciding factor regarding the method to be adopted is ordinarily the physical nature of the solids to be treated.[1]

[1] See p. 286 for a further discussion of this point.

PRACTICAL CONSIDERATIONS IN LEACHING OPERATIONS

Introduction.—Having discussed briefly the theory of dissolution let us now consider some of the ways in which the dissolution of solids in liquids is accomplished in actual practice. The problem is evidently to bring continually fresh solution in contact with the solid particles; that is, different solution, for though it may not be any less concentrated with the solute than the solution it displaces from around the particles still the mere friction of this displacement serves to reduce the thickness of the film surrounding it. Therefore it would seem that the most efficient way to accomplish this is to move the solid through the solution or the solution through the particles or to move both in opposite directions while in contact with each other.

The different systems of leaching may be listed as follows: (1) Leaching by percolation which means flowing the solution past the stationary solids and separation of the solution thus obtained from the residues or undissolved material. (2) Leaching by agitation in which dissolution is obtained while the solids are held in suspension in the solvent and a certain amount of relative motion is maintained and separation of the solution later by decantation or filtration. Each type listed above will be discussed showing the equipment involved, comparative costs, and its applications in industrial problems.

Leaching by Percolation.—This is a well-established method of leaching in which the material to be leached is placed in a tank equipped with a false bottom. The solvent is added at either the top or bottom of the tank and allowed to percolate through the material, being removed when it has passed through it. These tanks are usually arranged so that a counter-current system is employed, the new solids being added to the last tank and the weak liquor to the first and pumped successively from one tank to another till it reaches the last almost saturated. The dissolution of the solid material will have been completed in the early washes, the later washes being merely washing or displacing the stronger liquors retained by the solid material.

This process is well suited to cases where the material is porous and sandy or where the solid material is difficult to handle by other methods but is not applicable to materials which tend to pack into impervious masses. It has been found in practice that regularity in the size of the particles and not their size is the chief factor governing good percolation in such tanks, the idea being that where the particles are of all sizes the small ones pack in the openings between the larger ones thereby clogging up the channels. On the other hand, extremely fine material can be leached economically by percolation if it is preceded by a classification of the material so that the particles in any one vat are practically all of the same size. This then involves another process, namely of classification, in treating most materials and two processes of dissolution, one to treat the very fine sludge and another to treat the sandy matter. For this reason the use of percolation in leaching has been rather restricted in its application.

Metallurgical Practice.—Leaching of sands by percolation was very widely used in the United States up to a few years ago in the cyanide treatment of gold ores, and is still largely used in the Transvaal. The reason for its decline was the successful development of processes for the treatment of slimes at lower costs which resulted in the plants treating all their material as slime instead of separate treatments as before for sands and slimes. The question of removing the leached material from the tanks is handled in a number of different ways.

Sluicing is the cheapest where water and ample disposal room are available while shoveling or mechanical excavators are used where they are not. Percolation leaching is done on ores crushed as fine as 100 mesh and as coarse as 1 inch.

Diffusion Process in Beet-sugar Manufacture.—In leaching beet sugar the percolation method is used universally. It would be very difficult to handle this material in any other way and the development of labor-saving machinery has reduced operating costs tremendously. A battery is made up of 12 diffusors or cells. A diffusor has a capacity of about 42 to 86 bu. and its height is about twice its diameter. A grating made up of a perforated sheet-iron plate holds the sliced beets in the diffusor and when the dissolution is complete they are discharged through a discharge gate in the side near the bottom of a diffusor, being charged into the cell through a cover over the top of the diffusor.

The dissolution is effected by continuous counter-current leaching, the liquors being fed in at the bottom of the diffusors passing through the beets to the next, etc. Before placing a diffusor in the system therefore it is first necessary to drive out all the air. This is done by opening an air cock in the top of the tank and feeding the juice from the preceding diffusor at the bottom. When the juice reaches the level of the top of the diffusor the air cock is closed and the direction of the flow reversed. When the system has become balanced a calculated amount of the concentrated juice will be withdrawn from the system before the juice is run through the fresh charge.

The solution from each diffusor passes to heaters operated so that the temperature of the liquor rises from 35°C. for the juice leaving the sliced beets getting the fresh water to 70 to 75° for the strong liquors leaving the last diffusor containing the fresh beet cossettes. The amount of water required is 1.2 to 1.5 the weight of the beets producing 48 to 55 liters of juice per hectoliter of beets. The exhausted beets contain 0.5 per cent of sugar and about 95 per cent water, the original content having been about 15 to 18 per cent sugar and 75 to 86 per cent water or 94 per cent extraction.

The mechanism of diffusion of the sugar through the membranes surrounding the cells which contain the sugar depends upon the action of osmosis. There is a tendency for the sugar particles to pass through the membrane and into the surrounding solution (exosmosis) but there is also a tendency for water to pass from the solution through the membrane to the cell (endosmosis). These two counter actions will reach a balance when the strength of the solutions within and outside of the cells are the same. Therefore to obtain a complete extraction the solution outside the cells must be made progressively less concentrated. (The sugar in the beet being in solution from the start, the problem is not one of dissolution but of washing the solution from the solids and agitation is therefore not necessary.) The membrane around the cells of the beets as added to the diffusor, however, is covered with a film of protoplasm through which the diffusion is very slow and the solution is heated in order to coagulate this and thereby render it permeable.

Preparation of Organic Extracts.—Tanning extracts, etc. are usually prepared by leaching tan bark, etc. by percolation. Here as in the case of the diffusion process in beet-sugar manufacture there is no dissolution but from the start the process is a process of washing only. This offers no material difference in the procedure but limits the strength of the strong liquors to the strength of the original solution of this material in the plant cells. If the material were present in a crystalline form it would be possible to obtain saturated solutions from the percolation vats. This is far from being the case, however, in both the beet sugar and tanning processes.

Special Percolation Operations.—One type of percolation apparatus used largely in the cream-of-tartar industry is a semi-cylindrical perforated sheet-iron basket placed in a tank equipped with paddles. The crude argol is placed in this basket and the tank filled with water so that it covers the contents of the basket and the solution rotated by the paddles. As far as leaching is concerned it gives an excellent recovery and is a decided attempt at positive selective motion of the liquid through the solid but the power consumption is much higher than other types of leaching apparatus. It also has not the advantage of ordinary percolation over dissolution by agitation, in that the liquors are not clear after the agitation and they must be clarified in some way and furthermore it is difficult to apply the counter-current principle in this type.

In dissolving electrolytic sodium hydroxide as shipped in drums ordinarily some holes are punched in the drum and it is hung in the upper part of the tank of water. The dissolution will be fairly rapid due to the fact that the concentrated liquors being heavier than water sink, thus causing a circulation. This method is ordinarily slow and unless the material goes into solution very rapidly would not be applicable.

Leaching by Agitation.—One of the most common means of leaching is to agitate the material in the solute till the dissolution is complete, following this operation by a separation of the solids from the solution. There are two main types of apparatus for leaching by agitation. The first keeps the particles and solution in motion relative to each other and second merely holds the particles in suspension. (In quoting power consumption of machines the basis used in comparison is the power required to get equal extractions from equal tonnages of the same material.)

The paddle agitator (without baffles) consists merely of arms placed on a vertical shaft inserted through the top of a tank. It is rotated by any suitable motive power (this type is very seldom used without baffles but in order to bring out the faults of this class better we will discuss it briefly).

The only result of such an apparatus is to set the entire solution to rotating with the paddles. Fairly heavy particles will settle out from the solution very readily and after a very short time the entire system will come to equilibrium, the solution and solids and paddles all revolving at approximately the same speed and the agitation ceases, becoming only that due to retardation on sides and bottom of tank. The natural improvement is to place along the sides of the tank and between the paddles some stationary horizontal baffles.

The Baffle Paddle Agitator.—We now have a very violent agitation if sufficient baffles are used and the speed is about 10 to 15 r.p.m. The power consumption is 8 to 12 hp. for tanks 20 by 10 ft. This is due to the fact that here any relative motion of the particles through the solution is incidental to the setting up of violent eddy currents in the system. We are setting both the solids and solution in motion in the same direction so far as the mechanical action of the paddles is concerned, in direct violation of the principles already laid down. Furthermore in the space between the paddles there is no agitation and the power is consumed in that cycle in merely setting the solution in motion as in the ordinary paddle agitator though to be sure it does serve to bring the solution in those intervals to the agitating zone around the baffles but any point in the tank where the solution is not being agitated is going to retard the speed of dissolution due to the fact that it offers an opportunity for the heavier particles to settle out to the bottom

or to become stratified in the solution. If they do settle out in this interval there is no positive means of raising them and that brings up the last objection to this type and that is that it has been found in practice that there is not an even distribution of solids through the volume of a paddle agitator there being a much greater concentration in the lower layers than in the top.

Propeller Type.—This type consists merely of a propeller revolved in the bottom of a tank so as to throw the solution and solids up to the top of the tank where they are then sucked down by the propeller and shot up again. Baffles are placed along the side of the tank to counteract the slight tendency of the solution to rotate with the propeller. The criticism of this type is the same as with the baffle-paddle type. We start off as though what we actually wanted to do was merely to set up a lot of eddy currents of solid and liquid, whereas what we really want to do is to set up a selective current of the particles in the solution. The advantage it has over the baffle-paddle type is that there is a positive means of raising any particles tending to settle out and keeping them in motion and the distribution should be very good.

A combination of the propeller and paddle-agitator is got by inclining the bottom set of paddles so as to keep an upward motion of the solution in the tank.

Air Agitation.—This is the simplest of all agitators in many ways, the cheapest and simplest to install and the most easy way to waste air. It consists of a perforated pipe placed in the bottom of the tank and connected to a compressed air line. It gives a violent agitation and prevents very effectively settling from taking place. The efficient use of the air requires that it be evenly discharged over the entire area of the tank and in as small bubbles as possible. This is impossible to do and in most cases the bubbles are very large and are restricted over a small portion of the tank due to the holes having become clogged during the intervals when the machine was not in operation. Here again the same objection is raised that there is no attempt made to take a positive action to guarantee a relative motion between the undissolved solids and the solution. We practically try to raise the entire tank full of materials all the time, whereas in a latter paragraph we will mention other types of agitators which use compressed air to raise selectively the solids and naturally their power consumption is a very small fraction of that used by this type.

Dorr Agitator.—This is an ordinary tank in the center of which is a vertical shaft to the bottom of which are attached two arms with rakes, so that when the shaft is revolved they will rake any solids which settle out towards the center. This shaft in the center is hollow and has openings at the bottom and top and it is made into an air lift by a compressed-air pipe leading in at the bottom. To the top of the shaft are attached two perforated launders which revolve with the shaft and any material coming up through the center of the air lift flows into them and will be distributed over the entire surface of the solution in the tank.

If we place in a Dorr agitator a solid suspended in water the solids will of course tend to sink to the bottom as the arms revolve at such a low rate that there is practically no agitation by them. In sinking they pass through the solution and thus the film of concentrated solution is continually washed away. When they reach the bottom the rakes take them to the center of the tank and deliver them to the air lift which shoots them up to the launders where they are distributed over the surface of the solution and allowed to percolate through the liquid or solution again. Here

we see that there is a real attempt made to raise the solids selectively and let them percolate through the solution and to raise them again when they settle out. The solution elevated by the air lift will not be solids only but a pulp mass whose density above the average density depends upon the vigor with which the agitation is conducted.

There is also a violent agitation caused in the central tube in elevating the liquid so that the air consumed here is for agitation as well as raising the solids. It is found that there is a tendency for the coarser material which requires a longer treatment than the finer sands to collect in the agitator so that it received from one to four times the treatment that the more flocculent material obtains. Total power consumption is about 1.5 horsepower for a tank 20 by 10 ft.

Pachuca.—This type of agitator, similar to the Dorr in what it attempts to accomplish, has in recent years been almost completely replaced by other types. It consists of a very high tank with a conical bottom. Reaching from the top of the tank to the center of this cone is a pipe to be used as an air lift as in the Dorr, the idea being that as the particles sink they strike this conical bottom and slide

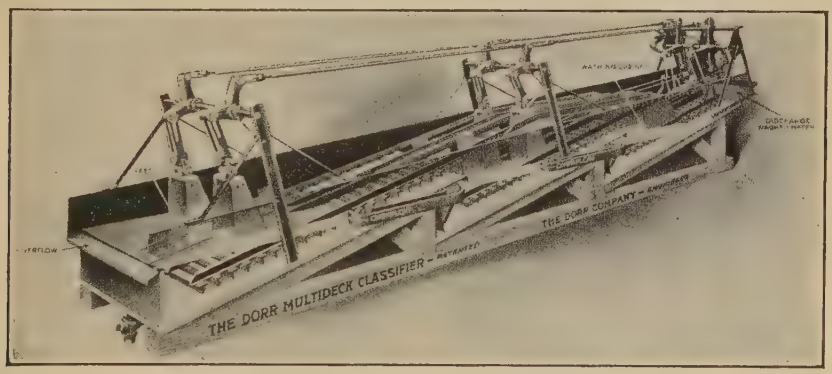

FIG. 6.—Dorr multi-deck washing classifier.

to the apex of the cone where they encounter the air lift. Here they are raised to the top and are distributed over the surface of the solvent as in the Dorr and again sink to the bottom.

The difficulty with this agitator is that the solids build up on the sides of the cone and then all slide down at once causing the air lift to become clogged or at best to work very irregularly. In order to increase the slope of the cone, the tanks are very high to get large capacity causing great expense in erecting and difficulty in placing under cover.

Dorr Classifier.—The Dorr classifier is being used in a number of operations for washing granular material free from value-bearing solutions or even directly for dissolving easily soluble materials. The multiple-deck Dorr classifier, especially designed for this work, is made up of a number of inclined settling boxes as shown in Fig. 6 in each of which is a set of rakes operated by one mechanism which move the solids settling out along the incline to the next compartment. The material to be washed is fed into the first compartment while the wash water is fed into the third and flows by gravity through pipes on the side connecting this compartment to the second and then the first where it is discharged. The height of the solution in the compartments is such that a draining period is allowed for every wash.

The movement of the rakes allows sufficient agitation to bring the washes in good contact with the crystals and the speed may be simply regulated to give sufficient time for the diffusion of the soluble matter from the interstices of the solids by the feed and rakes. Besides this it is a continuous operation requiring practically no attention.

Akins Classifier.—This machine is designed to accomplish separation in the same way as the Dorr classifier. In it the intermittently acting rakes are replaced by a continuously moving helix, the flights being interrupted at intervals to allow washing of the solids and return of the liquids.

Other Classifiers.—These include the Ovoca, using a continuous double helix; the Esperanza, using a continuously moving belt or chain equipped with rakes or paddles, and various other combinations of rakes and paddles, screws, etc. all based on the same principle. For definite particle-size classifications screens are more efficient than classifiers.

Continuous Counter-current Decantation.—This system is being quite generally used in the chemical world to replace the older intermittent agitation and decantation washing systems as well as more modern filter installations. It has as its basis the application of the Dorr thickener to remove suspended solids contin-

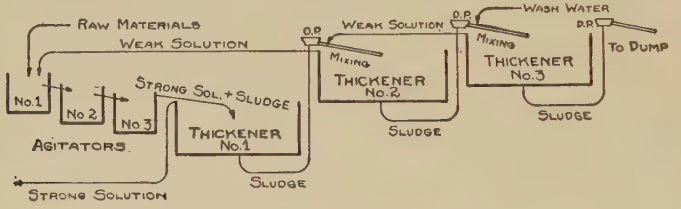

FIG. 7.—Typical counter-current decantation.

uously from solutions as they settle out. The Dorr thickener as shown on p. 280 consists of arms equipped with plows attached to a central shaft in the settling tank so that as these arms rotate they rake the solids which have settled out to the central solids discharge. The clear liquor overflows into the launder.

The power consumption of such a machine is less than ½ horsepower for mechanisms, in tanks up to 40 ft. diameter. The labor required by six thickeners is one-half of one man's time per shift and on larger installations will be proportionately less per thickener. The size tank required is the theoretical area required to settle out the solids continuously which is considerably less area than required by an intermittent system due to the fact that the time-consuming period of decanting and removing the solids is done away with by continuous operation.

In Fig. 7 is shown a typical continuous counter-current decantation installation together with the continuous-agitation system as it would be applied to the manufacture of phosphoric acid from phosphate rock and other processes which is a representative leaching problem.

The wash water is added in thickener No. 3 and the overflow from No. 3 goes to No. 2 and hence to the agitator to be used in making up the original solution. The solids settling out in No. 1 thickener are pumped from it to No. 2 by means of diaphragm pumps and the solids from No. 2 to No. 3, etc. By this system therefore, the solids are progressively coming in contact with weaker and weaker solution and as discharged by the last thickener will contain practically no values.

The percentage of extraction obtained by an installation such as this can be readily computed having the following data. First to what final density will the solids settle,

i.e., the ratio of water to solids. Second the strength phosphoric acid desired and third the amount of acid to be produced and amount of solids residues.

The first condition will control the number of thickeners required. Theoretically any washing efficiency up to 100 per cent can be obtained by this method though

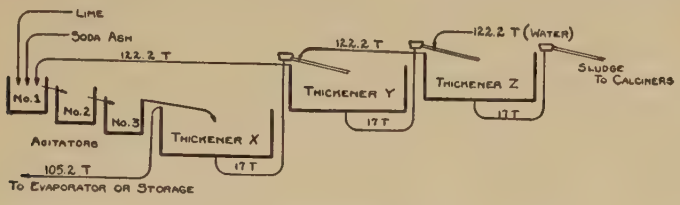

FIG. 8.—Counter-current leaching of sodium hydroxide.

practically it may not be feasible where the amount of solution carried from thickener to thickener by the solids is very great. The methods of computing the efficiency of such a system is shown in the following problem.

Conditions Assumed: From Fig. 8.

1. Ten tons NaOH produced per 24 hr.
2. Overflow from thickener X to have approximate specific gravity 1.1 (190 lb. NaOH per ton).
3. Yield of NaOH or "causticity" of finished product 92 per cent.
4. Time of agitation 2 hr.
5. Lime mud discharged from each thickener with approximately 1 part water to 1 part solids by weight.

Calculations:

Let X, Y and Z equal tons of dissolved NaOH per ton of water in the respective thickeners.

Equating total tons NaOH out of and into each thickener, we have:

1. $105.2X + 17X = 122.2Y + 10$ tons (NaOH)
2. $122.2Y + 17Y = 17X + 122.2Z$
3. $122.2Z + 17X = 17Y$

Solving:

$X = 7.32Y = 0.0948$ tons $= 189.6$ lb. NaOH per ton.
$Y = 8.20Z = 0.0129$ tons $= 25.8$ lb. NaOH per ton.
$Z = 0.00158 = 3.16$ lb. NaOH per ton.

Conclusions:

The 105.2 tons going to storage will contain at 189.6 lb. NaOH per ton or 9.974 tons NaOH.

This amount is 99.7 per cent of total 10 tons produced.

The Dorr classifier and the Dorr thickener it will be noted are designed to handle two different types of material. The classifier handles quick-settling sandy material which does not require long-washing periods while the thickener is best adapted to handling slow-settling slimes which require longer periods of detention in the washes.

Centrifugals.—A method sometimes used for removing the solution from solids is washing them in centrifugals. This method is of course expensive both in labor and power and is only resorted to in special cases. This method is used in washing the film of syrup from the crystals in sugar refining. The sugar and syrup are first spun for about 1 min. to remove the greater part of the syrup and then an extremely small wash applied by automatic washers. (It is of interest

to note distilled water is used in washing refined sugar to keep down the ash.) The wash need not be great since the ratio of water to solids is so very small. Where the product is subject to bacterial action but not affected by moderate heat, a centrifuging with steam both washes and sterilizes.

Wet Grinding.—Dissolution by grinding the solids in the solvent is a method seldom used in the chemical world but almost invariably in the metallurgical world on like problems. Where the solvent is neutral or non-corrosive the dissolution can be greatly accelerated by wet grinding at no greater cost. It offers distinct advantage in reducing dust and also offers great possibilities in lowering the cost of grinding.

Conclusion.—It is impossible to make any definite statement as to which of the various types of dissolution apparatus is to be regarded as the best due to the fact that each particular industry offers special problems and each material must be handled in a slightly different manner. However, it is perfectly evident that the chemical industry is reaching a point where it is necessary to introduce labor and power-saving machinery where in the past tonnages had not warranted it, and the process of dissolution and washing being a process which is practically universal among chemical plants will undoubtedly be studied and new machinery developed which will further increase its efficiency.

SECTION X

EVAPORATION

By Otto Mantius[1]

Bibliography.—Evaporators are used in such great numbers in all branches of the chemical industry that one would expect a considerable literature on this subject, giving data and instructions regarding design, installation and operation. As a matter of fact there is very little to be found in the form of textbooks. There are really only two books: F. Hausbrand, "Evaporating, Condensing and Cooling," an excellent theoretical treatise with some practical information; and Foster, "Evaporation," which is merely a collection of older treatises on thermodynamics, and fundamental evaporator patents. Besides, Prof. E. W. Kerr has tabulated and analyzed the results of some well-conducted tests with experimental evaporators in two pamphlets published by the Louisiana State University at Baton Rouge, La., *Bulletins* No. 138 and 149, and also with several types of sugar evaporators in a paper presented at the meeting of the American Society of Mechanical Engineers, New Orleans, La., April, 1916.

A series of articles, "Studies in Evaporator Design," by W. L. Badger, has also been published in 1920 and 1921 in *Chemical and Metallurgical Engineering*. Badger gives experimental data with special reference to the horizontal-tube evaporator.

History.—The art of evaporation by direct heat is naturally a very old one, and was used for the manufacture of drugs and pharmaceutical products by the Egyptians as described in the "Papyrus Ebers," 1500 B.C. Sugar and salt were recovered by the same process in the sixth century. The application of direct heat requires only a simple apparatus, but the fuel economy is low, and frequently the liquors are injured by the high boiling temperature. With the development of the cane- and beet-sugar industries, large quantities of water had to be evaporated, preferably at low temperature, and it was found that direct-heated evaporators were entirely inadequate for this work. In the early part of the nineteenth century, Howard and Roth designed and operated the first vacuum pans with condenser, and in 1829 Pecqueur constructed the first multiple-effect evaporator, using as heating surface hemispherical copper bottoms which were placed on top of each other. In 1845, N. Rillieux built the first multiple-effect with horizontal tubes; a few years later Robert constructed the first vertical-tube apparatus. The construction of all modern evaporators is based on these two fundamental types of machines as will be shown in a later chapter. Between 1880 and 1890, a new development took place which led to the introduction of the so-called film evaporators of Claassen, Yaryan, Lillie, Kestner, and others.

Theory of Evaporation.—Evaporation is the art of removing a solvent by application of heat and concentrating the solution until the dissolved substance separates or reaches the desired density. For instance, brine is concentrated

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until the salt crystals separate, and glucose or maltose solutions are concentrated until they reach the consistency of syrup. The main object is the recovery of the solids in a more or less concentrated form, and the solvent is wasted in practically all cases.

The various methods of evaporation may be divided into the following classes:

First: By direct contact between heating medium and liquid:

(*a*) Evaporation by waste gases and heated air.

(*b*) Evaporation by the rays of the sun (solar evaporation).

Second: By indirect contact between heating medium and liquid through metallic surfaces:

(*a*) Evaporation by direct heat produced by coal, gas or oil.

(*b*) Evaporation by indirect heat conveyed by steam, hot water, oil or electricity.

All the methods mentioned above are used in the chemical and allied industries for the concentration of various liquors, but there is no doubt that the heating with steam gives the best results, and most of all, the highest efficiency. In practically all cases, water is the solvent, and as early as 1829 Pecqueur found that it was possible to use the vapor produced in one evaporator as a heating medium in the second apparatus working at a lower pressure and temperature. The vapor produced in this body can be used again in the next evaporator, and so on as long as there is sufficient temperature difference between vapor and liquid to cause boiling of the liquid. A pound of coal with about 13,000 B.t.u. will produce in a steam boiler about 8 lb. of steam with 9,100 B.t.u., an efficiency of 70 per cent. This amount of steam, however, will evaporate again about 7 lb. of steam in one evaporator (single-effect), 14 lb. in two evaporators (double-effect), 21 lb. in three evaporators (triple-effect), and so on. In a combination of a boiler and triple-effect, 1 lb. of coal of 13,000 B.t.u. would therefore produce 29 lb. of steam of about 33,000 B.t.u., an efficiency of over 250 per cent. This is a very important fact, and the low cost of two of our most essential staple articles of food, sugar and salt, are due to the high efficiency of multiple-effect evaporators; also soda pulp could not be manufactured in competition with sulphite pulp if the soda in the weak wash waters could not be recovered economically by multiple-effect evaporation.

Waste Gases and Heated Air.—This method consists of bringing the hot air or gases in intimate contact with the more or less finely divided liquid. For instance, the liquor is placed into large shallow pans and the furnace waste gases are conducted over the top of the pan. Even under favorable conditions, the contact surface is small and the evaporating capacity is low, *i.e.*, from 2 to 3 lb. of water per hour per square foot, depending on the temperature of the gases. A much larger contact surface and better efficiency is obtained in the so-called Carlson Evaporator, which is used in the sulphate-pulp mills and where a large surface is exposed to the waste gases by leading these gases through a series of revolving discs which are partly submerged in the liquid. To the same class belong the spray evaporators, where the liquor is divided into fine particles by means of a horizontal disc rotating at a high speed, or by discharging it under high pressure from spray nozzles. This method is used mostly for drying purposes, and will be described in more detail in the next sub-section: "Drying." In all cases, the evaporation is effected by lowering the temperature and increasing the humidity of the waste gases or hot air; consequently, large volumes are required and the efficiency is low.

Solar Evaporation.—Nature has given us in the sun's rays a practically inexhaustible supply of heat, but it can only be utilized to advantage in warm dry

climates. Large quantities of salt are manufactured by this method from sea water in southern California, South America, Portugal, France, Spain and Italy. Salt plants in Germany and Austria concentrate weak brines by pumping them repeatedly over high towers made of wood and filled with thorn brushes (Gradierwerke). During the last few years, weak potash brines have been concentrated in the Western deserts by solar evaporation with good success. This process, however, requires large exposed surfaces, as the evaporation amounts to only about 1½ lb. of water per square foot per day in Nevada, and about 1 lb. in Brazil.

Evaporation by Direct Heat.—Next to solar evaporation, this is probably the oldest method of concentrating liquors. Open pans with bottoms of various shapes to increase the heating surface are still used to a large extent in the chemical industries on account of their low first cost and simple construction. Sometimes straight or corrugated flues have been added to increase the heating surface, but on account of the low factor of heat transmission between hot gases and a boiling liquid, the evaporative capacity is small. Even with a well designed furnace, the evaporation will not exceed 5 lb. of water evaporated per square foot per hour, and under average conditions will not be more than 3 lb., the same as in standard boiler practice. The efficiency is low because the gases generally escape at a high temperature. One pound of coal will usually evaporate from 4 to 6 lb. of water and only rectangular shaped salt pans (grainers) which are from 40 to 60 ft. long, and where the flues are arranged in four or more passes have actually shown an efficiency of 10 lb. of water evaporated per pound of soft coal. Fuel oil and natural or artificial gas will give similar results to coal if the difference in B.t.u. is considered. Waste gases from boilers and furnaces may be used but the evaporative effect is low and will not exceed 2 lb. per hour per square foot depending on the temperature of the gases.

It has been suggested quite frequently to combine a direct-heated tubular pan with a multiple-effect evaporator in order to save the cost of the boiler. In practically all cases these attempts have resulted in complete failures because of the difficulty of constructing and operating such an apparatus, and also for the reason that the first cost of the direct heated evaporator is much higher than the combined cost of a standard boiler and evaporator.

Evaporation by Steam.—Saturated steam is by far the most efficient heating medium, and is used in the chemical and allied industries at various pressures in single- and multiple-effect evaporators, working under pressure or vacuum. Superheated steam is not suitable for evaporation of liquids in large quantities as it acts as a gas and therefore requires very large heating surfaces.

Hot water as a heating medium is used for very low and also for high temperatures. In the first case, the water is circulated through coils or jackets at low temperature under ordinary pressure to prevent burning of the liquid to be evaporated. In the second case, the water is heated under high pressure for the purpose of evaporating liquids which have a very high boiling point. The amount of water in circulation is small to prevent serious accidents in case of explosions.

Oil is used frequently instead of water under high pressure to eliminate the dangers from explosions. Both these methods of evaporation are expensive in first cost and operation, and are only applicable for special cases. Electricity as source of heat is only used in laboratory apparatus; for commercial purposes it may be used where fuel is not obtainable, and electricity can be produced by cheap waterpower.

Heat Transmission.—In order to transmit heat from one body to the other there must necessarily exist a temperature difference, and generally speaking the capacity of an evaporator depends on the temperature difference between the heating steam and the boiling liquid.

A number of conditions influence the heat transmission.

1. Material, thickness and cleanness of heating surface.
2. Temperature difference depending on temperature of heating steam and boiling liquid, also density, viscosity, total depth and circulation of the liquid.
3. Velocity, distribution, density and quality of the steam.

The heat to be transmitted through a metal surface has to overcome three resistances:

1. The entrance from the steam into the metal surface.
2. The passage through the metal.
3. The exit from the metal into the boiling liquid.

These factors have been determined by numerous experiments, and the average results are given in L. S. Marks' "Mechanical Engineers Handbook," pages 303–306, as follows:

k_1 = 2,000; denotes coefficient of conductance of steam film.
k_2 = 1,000; denotes coefficient of conductance of boiling liquid film.
K = coefficient of thermal conductivity of the plate for a thickness of 12 in. (see L. S. Marks: Tables No. 19, 20, 21).
K/b = coefficient of conductance of plate for a thickness of b in inches.
R = resistance to the flow of heat through films and plate.
$k_0 = 1/R$ = total sum of conductances.
t_1 and t_2 = temperature of steam and boiling liquids.

All factors are given per degree Fahrenheit, and Q is the total heat in B.t.u. transmitted per hour per square foot.

For an evaporator where steam and boiling liquid are separated by a metal surface, $Q = k_0(t_1 - t_2)$, and $R = 1/k_1 + (1/k_2) + (b/K)$.

For a copper tube, $K = 200$. The thickness of the tube is usually $\frac{1}{16}$ in.; therefore $b = \frac{1}{192}$, and with a steam pressure of 5 lb. and a vacuum of 26 in., the total temperature difference would be 102°F. Under these conditions, $R = (\frac{1}{2,000}) + \frac{1}{1,000}) + (1/(192 \times 200)$, $Q = 1/655$. $Q = 655 \times (t_1 - t_2) = 66{,}810$ B.t.u.

This equation shows that with other conditions equal, the thickness of the tube has very little influence on the transmission of heat, and with a copper tube $\frac{1}{8}$ in. thick, the amount of heat transmitted would be 66,300 B.t.u. For an iron tube $\frac{1}{16}$ in. thick, and $K = 35$, Q would be 61,812 B.t.u. The small theoretical advantage of copper over iron is also true in actual practice, except that usually iron tubes are more easily corroded than copper tubes, and the corrosion forms an insulating film on the iron which greatly reduces the heat transmission. For practical purposes iron of the same thickness will transmit about 80 per cent of the heat transmitted by copper.

For lead tubes $\frac{1}{4}$ in. thick, and $K = 20$, Q would be 40,086 B.t.u.

Scale will exercise considerable influence on the amount of heat transmitted. Correct data for K are not known in this case, but may be taken the same as boiler scale, which will average about 1.50. The total resistance will be the sum of the resistance of the clean tube and the resistance caused by the scale. Therefore, for a $\frac{1}{16}$-in. copper tube coated with a $\frac{1}{32}$ in. lime scale, $R = \frac{1}{655} + (1/(384 \times 1.50)$, $= \frac{1}{300}$; Q is 30,600 B.t.u., that is about 50 per cent of the heat transmitted through a clean copper tube. This reduction in heat transmission is somewhat higher than in actual practice, which is probably due to the fact that in most cases scale is not very dense and will permit direct contact between the metallic surface and the boiling liquid. Incrustations of salt and organic matter have a similar influence, and it happens

frequently that the capacity of an evaporator is reduced to one-half or one-third by an apparently thin coating of the solids on the tube surface.

For water at rest, k is about 100, but the conductance will increase rapidly with the increased velocity. Tests made by Clement and Garland (*Bull.* No. 40, Engineering Experiment Station, University of Illinois) show values for k of 730 at 1.45 ft. per second, and 2,500 at 17.13 ft. per second, which is considerably higher than the average conductance of boiling water. This fact is important for the construction of preheaters and surface condensers, and various experiments have given the following relations (*Trans. A. S. M. E.*, Vol. 32):

Ser, $k = 520\sqrt[3]{v}$; Josse, $487\sqrt[3]{v}$; Stanton, $340\sqrt[3]{v}$; Clement and Garland, $270\sqrt[3]{v}$; Hagemann, $282\sqrt[3]{v}$; Allen, $220\sqrt[3]{v}$; Orrok, $308\sqrt[3]{v}$ where v denotes the velocity of the liquid in feet per second.

As stated before, the total quantity of heat transmitted depends on the tempera-

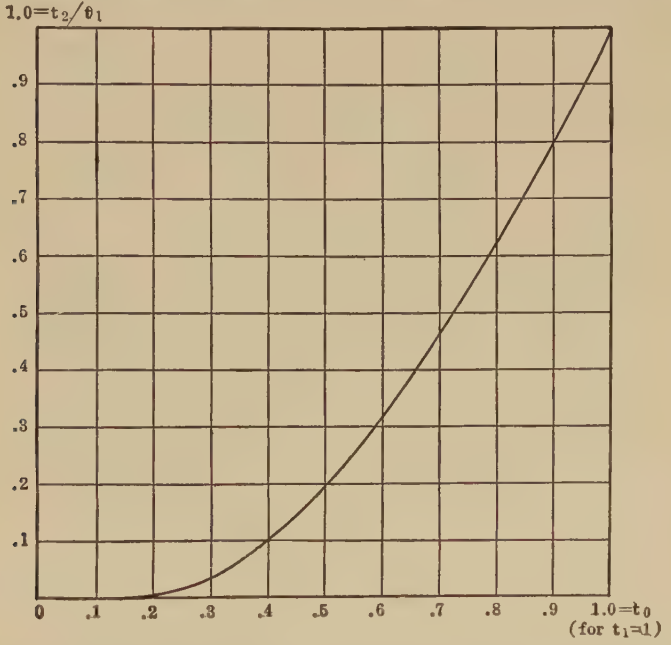


FIG. 1.—Heat transmission constants.

ture difference between the heating medium and the liquid. In ordinary evaporator practice, this difference is fairly constant, and can readily be determined by thermometers. In preheaters and condensers, however, the temperatures change and the average temperature difference will be $t_a = (t_1 - t_2)/n(1 - \sqrt[n]{t_2/t_1})$ (taken from E. Hausbrand). t_1 denotes the initial and t_2 the final temperature difference; n is a factor varying from 10 to 100 or higher, and may be taken arbitrarily, depending on the exactness of the result wanted. The results of this equation have been plotted in Fig. 1.

For instance, for $t_1 = 20°$ and $t_2 = 12°$, $t_2/t_1 = 0.60$, and t_a (for $t_1 = 1$) $= 0.80$, therefore $t_a = 16°$.

The total sum of conductance K_o is commonly called the coefficient of heat transmission, and for steam at 212°F. and a copper tube $\frac{1}{16}$ in. thick, has been figured as 655. This amount will vary with the density of the steam, and empirical formulas have been proposed by various authors. Professor Kerr has determined K_o by a large

number of tests on experimental evaporators with copper tubes, and the average results are represented by the formula

$$K_o = 225 + 17{,}500D$$

where D is the density of the heating steam in pounds per cubic foot. For steam at atmospheric pressure (212°F.), K_o would be 925; for a vacuum of 20 in. which usually prevails in the steam chest of a fourth effect, K_o would be 450. Both these figures are much higher than the results obtained in actual practice.

The coefficient of heat transmission also changes with the density and viscosity of the liquor, and the reduction is usually in direct proportion to the specific gravity. Impurities and suspended matter will reduce the factor, and high velocity and good distribution of the steam will greatly increase the heat transmission which is evident in evaporators with properly proportioned steam coils which sometimes show exceedingly high capacities per square foot of heating surface. The heating steam always contains air and non-condensable gases which greatly reduce the heat transmission, as the air seems to form an insulating surface on the tubes. It is therefore of the utmost importance to provide efficient means of removing air and gases from the heating element. Steam gages will show the total pressure of steam and air in the steam chest, but the actual temperature is only due to the pressure of the steam and should therefore be determined by a thermometer and not from the gage pressure. According to Orrok, K_o varies as $(p_s/p_t)^2$ where p_s denotes the pressure of the steam alone, and p_t the total pressure ($= p_s +$ pressure of air).

The condensed steam forms a layer of water on the heating surface, and it is necessary especially in coils and horizontal tube evaporators that this condensate is removed as quickly as possible to avoid reduction in capacity.

Rapid circulation of the liquid and a low liquor level will greatly increase the capacity. A thin film will naturally cause a rapid evaporation, and it has been found that in all vertical-tube evaporators the highest capacity is obtained by keeping the liquor level at about one-third of the length of the tube above the lower flueplate. The upper part of the tube is then covered only by a thin film which evaporates very quickly. This method can, however, not be used in cases where salts are separated from the liquid, as these salts would naturally form a coating on the tubes. For further details regarding influence of liquor level see E. W. Kerr, *Bull.* 138 and 149, Louisiana State University.

It has been shown that the actual amount of heat transmitted will depend on a great many factors and will vary considerably with the type of apparatus and the kind of liquor. Outside of water distillation, the extreme limits in actual practice are probably 8 lb. per hour per square foot for electrolytic caustic liquor from 25 to 48° in a vertical-tube evaporator and 62 lb. evaporation per hour per square foot for malt extract from 5 to 30°Bé. in a rapid circulation film type evaporator with steam at 5 lb. and a vacuum of 28 in. Practical results of various liquids are given in later paragraphs, p. 375 *et seq.*

Evaporation Under Vacuum.—Boiling under reduced pressure has the following decided advantages: Increased temperature difference; lower initial steam pressure; low boiling temperature of the liquid; reduced steam consumption. An increased temperature difference means a higher capacity per unit of heating surface and therefore a smaller heating surface for the same output. A vacuum evaporator is, however, more expensive than the atmospheric type, as it requires a closed vessel with large vapor space and a condensing system.

A great economy is often accomplished by utilizing waste steam of engines and pumps instead of high-pressure direct steam, and the increased cost of equipment is frequently paid for in a few months by the saving in coal.

The reduced pressure or vacuum naturally lowers the boiling point of the liquid, and vacuum evaporation is therefore used to advantage in all cases where high boiling temperatures would injure the produce (sugars) or in cases where the boiling point of the concentrated liquor is so high that even direct steam would not give sufficient temperature difference to cause boiling (caustic soda). Where large quantities of cold liquid are to be handled, the low boiling point gives the additional advantage that the liquid has not to be heated to the higher boiling point due to atmospheric pressure.

Evaporation in Multiple Effect.—This process may be carried out under pressure or under vacuum. Multiple effects working under pressure are very rare, and are only used for water distillation. Vacuum evaporation in multiple effect is used extensively for all kinds of solutions, and forms, without doubt, the most important branch of this industry.

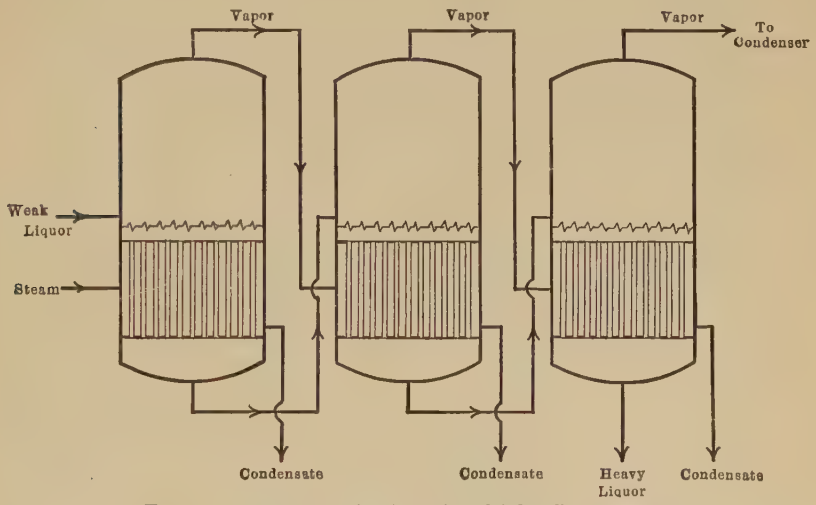


FIG. 2.—Diagrammatic view of multiple-effect evaporator.

In multiple-effect evaporating system, a number of evaporators are connected in series so that the vapor produced by the steam in the first unit is introduced into the steam chest of the second effect; the vapor of this evaporator is again passed into the steam chest of the third effect, and so on as shown in Fig. 2.

The latent heat of the steam entering the first effect passes through the heating surface and causes boiling of the liquid. The resultant vapor containing practically all the latent heat of the steam will give off this latent heat again to the liquor in the second effect, and so on. The actual work is done by the transfer of the latent heat of the original steam through the heating surface from one evaporator to the next until it is finally condensed in the condenser attached to the last effect. Theoretically, there is almost no limit to the number of times that this latent heat may be transferred; practically, however, the limits are given by the fact that it takes a certain temperature difference between the vapor on one side of the heating surface and the liquid on the other to cause boiling and produce again vapor. For plain water and very weak solutions, this limit of temperature fall is 10°F., and for other liquor, from 20 to 30°F. In the first case, the limit is a decituple effect, and for ordinary conditions a triple or quadruple effect.

It should be borne in mind that a multiple effect, where evaporators are connected in series, has the same capacity as a single effect, provided the initial steam pressure and vacuum are the same. However, the steam consumption is reduced in proportion to the number of effects installed and this increased economy is the only reason for installation of multiple effects, which are naturally much more expensive than single-effect evaporators. In some cases, evaporators are arranged parallel with individual steam connections and condensers, and it is evident that the capacity will increase with the number of evaporators, but the steam consumption will also rise in the same proportion. In the first case, the total temperature fall T has been divided into a number of smaller temperature falls, T_1, T_2, T_3, etc., and the evaporating capacity of the multiple effect will be $Q = K_o (T_1 + T_2 + T_3, \text{etc.}) = K_o T$, which is also the capacity of a single effect. In the second case, the capacity of each evaporator, $Q_n = K_o T_n$ and therefore the total output of all the evaporators equals the sum of all Q's, $Q = K_o T$.

Steam Consumption.—It is apparent that any evaporating system should have complete heat balance and that in all cases the amount of heat entering and leaving the evaporators should be the same. If we call

S = B.t.u. in steam admitted to steam chest,
V = B.t.u. in vapor going to condenser,
R_1, R_2, R_3 = B.t.u. lost by radiation in first, second and third effect,
W = B.t.u. in weak liquor entering evaporator,
H = B.t.u. in heavy liquor leaving evaporator, and
C_1, C_2, C_3 = B.t.u. in condensate discharged from evaporators.

The heat balance of a single effect would be

$$S = V + C_1 + R_1 + H - W$$

and for a triple effect

$$S = V + C_1 + C_2 + C_3 + R_1 + R_2 + R_3 + H - W$$

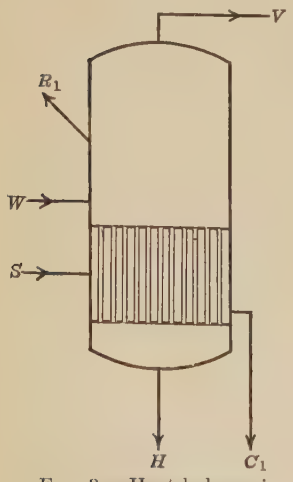


FIG. 3.—Heat balance in single effect.

These equations give the factors that determine the steam consumption of an evaporator and make it clear that all types of evaporators must have the same efficiency with reference to steam consumption provided the working conditions are identical. Therefore the claim of some manufacturers that their particular type of evaporator consumes less steam than other constructions is misleading as there can only be a difference in the losses by radiation, and with a good insulating covering these losses are so small that the difference is negligible.

In a single effect, the determination of V is simple, as it can readily be figured from the difference between the weight of the weak and heavy liquor, the temperatures and the vacuum. In a multiple effect, the problem is rather difficult as for the determination of V it is necessary to know the quantities of water evaporated in each effect. The evaporation in each effect is not the same, but will be smallest in the first, and largest in the last effect, on account of the fact that the liquor passing from one evaporator to the next will give off some of its heat (reëvaporate). This vapor added to the vapor coming from the first effect will naturally increase the amount of evaporation in the second effect, and so on. The exact amount of reëvaporation depends on

the temperature difference between the two evaporators, and these temperatures cannot be determined exactly beforehand. The case becomes even more complicated if the condensate of the second effect is discharged into the steam chest of the third effect to save the heat of reëvaporation. E. Hausbrand, in his book, "Evaporating, Condensing and Cooling," has given some excellent formulas and tables for various temperatures and percentages of concentration taking into account the amount of heat

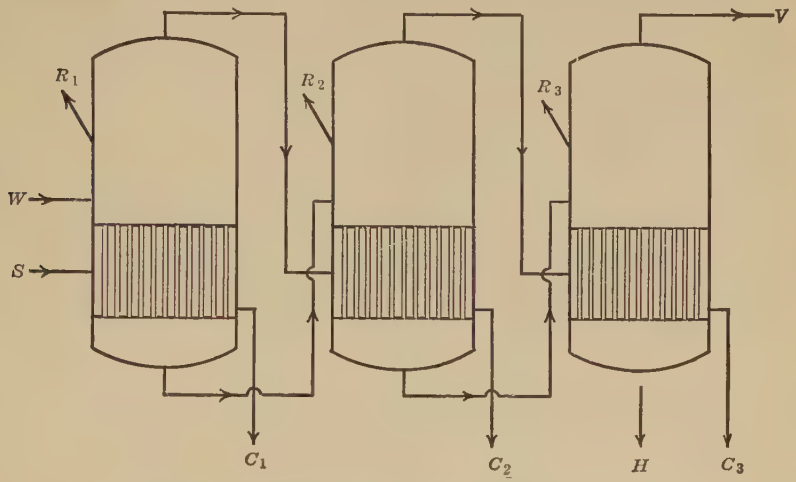


FIG. 4.—Heat balance in multiple effect.

recovered by reëvaporation of the liquor passing from one body to the next with the assumption that the condensate is discharged from each steam chest separately. The average results are:

Double effect; $V_1 = 0.466$; $V_2 = 0.534$
Triple effect; $V_1 = 0.300$; $V_2 = 0.329$; $V_3 = 0.371$
Quadruple effect; $V_1 = 0.216$; $V_2 = 0.243$; $V_3 = 0.256$; $V_4 = 0.285$

where V_1, V_2, V_3, V_4 are the amounts of evaporation in each effect, and the sum is the difference between the weight of the ingoing and outgoing liquors. These percentages are correct for a concentration of the liquor of 7 to 1, and V_1 will be about 8 per cent smaller for a concentration of 10 to 1, and 5 per cent larger for a concentration of 4 to 1. All these figures are based on the assumption that the liquor enters at the temperature prevailing in the first effect and allowances must of course be made if the liquor is warmer or colder.

In practically all cases, the condensate coming from the steam chest of the first effect is returned to the boiler; in a triple or quadruple effect a slight saving in steam may be accomplished by discharging the condensed water from the second steam chest into the third and from the third into the fourth steam chest. For average conditions the factors of distribution in this case would be:

Triple effect; $V_1 = 0.295$; $V_2 = 0.324$; $V_3 = 0.381$
Quadruple effect; $V_1 = 0.211$; $V_2 = 0.238$; $V_3 = 0.256$; $V_4 = 0.295$

The steam consumption will also depend on the amount of heat lost by radiation. For evaporators not covered by insulating material, the loss R will be 3 B.t.u. per degree Fahrenheit per hour per square foot of outside surface, and 0.5 B.t.u. per degree Fahrenheit per hour per square foot for apparatus that are well covered with magnesia or corrugated asbestos paper. The degrees Fahrenheit are equal to the

average of the difference between the temperatures prevailing inside the evaporators and the room. This loss by radiation should be added to the total amount required for evaporation.

The equation for steam consumption may now be rewritten as follows:

$$S = V_1 + W/L(t_1 - t)$$

where, S denotes pounds of steam entering first steam chest,

V_1 denotes pounds of evaporation, consisting of difference in weight between ingoing and outgoing liquor and loss by radiation multiplied by factor of distribution,

W denotes pounds of liquor entering the evaporator,

L denotes latent heat of steam entering steam chest,

t_1 denotes temperature of boiling liquid, and

t denotes temperature of liquor entering evaporator.

The concentration of 14,000 lb. of weak liquor at 70°F. to 2 000 lb. in a triple effect with steam at 5 lb. pressure and a vacuum of about 27 in. in the last effect, would require an evaporator with an outside surface of about $3 \times 800 = 2{,}400$ sq. ft., and the loss by radiation would be $R = 2{,}400(160° - 70°) \times 0.5/960 = 113$ lb., if 160° is the average temperature in the three evaporators. The temperature in the first effect would be 190°F., and

$$V_1 = (14{,}000 \text{ lb.} - 2{,}000 \text{ lb.} + 113 \text{ lb.}) \times 0.295$$
$$S = 3{,}573 + 14.6(190° - 70°) = 5{,}325 \text{ lb.}$$

If the weak liquor should have a temperature of 212° instead of 70°, then

$$S = 3{,}573 + 14.4(190° - 212°) = 3{,}541 \text{ lb.}$$

which shows plainly the importance of the temperature of the ingoing liquor. In cases where the liquors contain large percentages of solids these should be taken into account when figuring the steam required for pre-heating.

The foregoing figures have all been based on the usual practice of running the multiple effect "direct current," which means that the weak liquor enters the first effect and passes from there to the second, third and fourth effect. In some cases, however, it has been found advantageous to run "countercurrent" by feeding the liquor into the last effect and convey it from there by pumping to the previous evaporators. Naturally there will be no reëvaporation and the proportion of vapors will be distributed so that the first effect will have the highest and the last evaporator the lowest amount of evaporation, as the liquor has to be heated going from one effect to the other. This method of running countercurrent will save a small percentage of heating steam in case large amounts of cold liquors are entering the evaporating system, and has also other advantages: It will eliminate losses by entrainment which is frequently caused by the rapid reëvaporation of the hot liquor entering the next evaporator; it will save from 20 to 40 per cent of cooling water; it will boil and discharge the finished liquor at high temperature. Countercurrent operation will require, however, pumping of the liquor from one effect to the next, and this is frequently undesirable especially with chemical liquors that will corrode metals. Further details may be found in a paper read by H. K. Moore and published in "*Chemical and Metallurgical Engineering*," Vol. 18, p. 187 *et seq.*

A third method is "semi-countercurrent" where the weak liquor enters the second effect and passes from there to the third and fourth. The liquor of the last evaporator is pumped into the first, and discharged at high temperature (black liquors and packing-house tank waters).

In some industries, particularly in beet-sugar factories, vapor is taken from the first or second effect to preheat the raw juice, and this will naturally introduce complications in the calculation of the total steam consumption (see E. Hausbrand).

Recently another system of evaporation has been developed in Europe, in which vapors coming from the evaporator are compressed in a multi-stage turbo-compressor, and then returned into the steam chest at higher temperature. It is claimed that under certain conditions, the fuel consumption of a single effect is less than that of a quadruple effect. For detailed information, see an article by Carlsson in *Chemical and Metallurgical Engineering*, Apr. 13, 1921, p. 645.

Water Consumption.—With few exceptions, the vapors coming from the last effect must be condensed in some kind of a condenser, and the amount of water needed for this purpose is frequently a very important factor.

The quantity of cooling water can be determined by the following equation:

$$W = V_1 \times L/(t_1 - 10 - t)$$

where,

W denotes the quantity of cooling water in pounds,
V_1 denotes the amount of vapor coming from the last effect,
L denotes the latent heat of this vapor,
t_1 denotes the temperature of this vapor,
t denotes the temperature of the cooling water entering the condenser, and
$t_1 - 10$ denotes the temperature of the water discharged from the condenser.

Well constructed condensers will operate satisfactorily with a difference of 5°F. between the temperature of the vapor and the hot water; but it is safer to figure on a difference of 10° for actual operating conditions, and therefore this figure has been used in the above equation. The temperature t is known, and t_1 and L can be figured from the vacuum at which the last effect is supposed to work (see steam tables in Kent or Marks, also a short table, p. 48, this book). V_1 can be determined from the data given on page 365, covering factors of distribution.

Evaporation and Percentage of Solids.—In many cases only the amount of weak or heavy liquor and the specific gravity or percentage of solids of solutions are known. The determination of the steam and water consumption is, however, based on the actual amount of water evaporated, and this quantity may be figured as follows:

$$V = L(1 - l/h) = H(h/l - 1)$$

where V denotes total quantity of water evaporated in pounds,
L denotes amount of weak liquor in pounds,
H denotes amount of heavy liquor in pounds,
l denotes percentage of solids in weak liquor or its specific gravity minus 1, and
h denotes percentage of solids in heavy liquor or its specific gravity minus 1.

These equations may also be used to figure out various factors useful for evaporator practice:

$$H = L \times l/h;\ L = H \times h/l;\ l = h(1 - V/H)$$
$$H = V/(h/l - 1);\ L = V/(1 - l/h);\ h = l/(1 - V/L)$$

The factors of distribution mentioned on page 365, and the above formula will give the data to figure the percentage of solids or specific gravity of the solution in each effect of a multiple-effect evaporator. This is frequently important for the design and operation of evaporators, as it gives means for the determination of the boiling point, viscosity and crystallizing point.

LOSSES IN EVAPORATORS

Radiation.—The outside surfaces of all evaporators, vapor lines, and liquor lines should be covered with an efficient insulating material so as to reduce these losses to a minimum. This is of special importance when evaporators are used intermittently. Average figures have been given in a previous paragraph, "Steam Consumption."

Entrainment is always due to faulty construction or operation of the evaporator. Small particles of liquor are carried to the next steam chest or to the condenser with the vapor, and this can only take place where the vapor speed is too high. Catchalls and entrainment separators will reduce these losses, but evaporators should be designed and operated so that there will be no entrainment.

Foaming or frothing is common with a great many alkaline solutions; and evaporators used for the concentration of such liquors should be especially adapted for that purpose. In practically all cases, losses by foaming can be prevented by keeping the liquor level as low as possible, in which case the foam bubbles will be broken by coming in contact with the hot surfaces of the tubes that are not covered by liquor. This method of operation will also increase the capacity, as the liquor will be spread in a fine film over a large part of the heating surface. Catchalls and separators are necessary, and will save valuable material in case of careless operation. Uniform working conditions and constant low liquor level are very helpful.

Incrustations.—A great many solutions have the tendency to deposit a scale on that part of the heating surface which comes in contact with these liquors. Such a scale will always cause a considerable loss, not so much in efficiency as in capacity. Actual figures for the reduction in heat transmission have been given in previous paragraphs. Experience has shown that all substances which are less soluble at higher temperatures will form a hard scale on the tubes, while all salts that increase in solubility with rise in temperature will form incrustations that are easily soluble in water, and can therefore readily be removed by boiling. All liquors that have a tendency to deposit a hard scale on the tubes require an evaporator where the tubes can be cleaned mechanically.

Arrangement and Construction of Evaporators.—Local conditions must decide the very important question whether single- or multiple-effect evaporators should be installed. If exhaust steam is to be used only, the number of effects will depend on the quantity available, to which would be added the amount of exhaust steam coming from the pumps of the evaporating equipment proper. In cases where live steam has to be used, it is necessary to compare the saving in fuel with the additional expense for depreciation, interest and repairs of the evaporators. It is natural that for small quantities of liquor, a single or double effect is sufficient, and that only for large plants triple or quadruple effects should be chosen. Of secondary importance is the water consumption for cooling purposes in the condenser, but where water is scarce, it is always advisable to install a cooling tower.

The number of effects is often limited by the maximum temperature at which the liquors can be handled without spoiling them, and in some cases even for large quantities a single effect has to be used, as otherwise the boiling temperature would injure the finished product (gelatin, malt extract). The number of effects is also depend-

ent upon the available temperature differences between the heating steam and the vacuum and this temperature difference is frequently influenced by the increased boiling point of the solution (caustic soda, calcium chloride).

Evaporators are usually operated with exhaust steam at from 0 to 10 lb. pressure, and with live steam up to 30 lb. pressure, as a higher pressure will greatly increase the cost of the apparatus. The vacuum in the last effect depends on the amount and temperature of condensing water available, and for all smaller installations and ordinary liquors, a wet system condenser with from 25 to 26 in. of vacuum is the usual practice. For larger plants and for delicate liquors, it is important to have a high vacuum, which must be produced in a barometric condenser with dry-vacuum pump, or in a surface condenser in cases where it is not permissible to mix the vapors with the cooling water. Usually a vacuum of from 27½ to 28 in. is carried in these systems, and it is very seldom that a higher vacuum than 28 in. is required.

It has been shown under "Heat Transmission" that the temperature differences in a multiple effect are not evenly distributed for the reason that the factor of heat transmission increases and decreases with the steam pressure. In the earlier stages of evaporator manufacture, multiple effects have been built with various sizes of heating surface so that the temperature difference between each effect would be the same, claiming that this would give better efficiency and capacity. All modern evaporators are built with equal heating surface in each effect, as this will naturally reduce the cost of production, and it has been demonstrated under these conditions each effect will do its proper share of work. Only in cases where large amounts of extra steam are to be taken from one or more effects of a multiple effect, it has been found necessary to increase the heating surface in proportion to the extra amount of vapor to be produced. Such evaporators are frequently installed in beet-sugar factories where large amounts of extra steam are used for the preheating of the weak juice.

Material of Construction.—A good quality of close-grained cast iron should be used for all evaporator bodies as it will reduce the amount of leakage to a minimum. In all plate metal and copper construction, the large amount of rivets and seams are always a source of trouble. Recently steel-plate evaporators have been used with welded seams, but this method of construction has not been satisfactory for strong alkaline solutions. The heating surface mostly consists of tubes made of steel, wrought iron, charcoal iron, copper, brass, bronze, aluminum and lead. Evaporators have been built with aluminum shells and of glass enameled steel. There is practically no limit as to what material can be used for the construction of evaporators, and in some cases, steel and cast-iron shells have been lined with acid-proof brick in order to reduce the action of acid liquors on the metal shell. Evaporators have been built of solid lead with lead coils, and also of cast iron with homogeneous lead coating, and sometimes a separate lead lining. Special information on this subject will be given in later paragraphs.

VARIOUS TYPES OF EVAPORATORS

Jacket and Coil Type.—Figure 5 shows the construction of the first multiple-effect evaporator invented and patented by Pecqueur, and the heating surface consists simply of dished copper bottoms. The whole arrangement of the multiple effect is very simple, but naturally the capacity is small as such a construction is always limited in size. Figure 6 shows Howard's vacuum pan with jacket for the concentration of sugar juice. A similar pan was used by Roth for the same purpose.

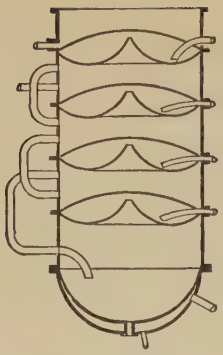


FIG. 5.—Pecqueur's original multiple effect.

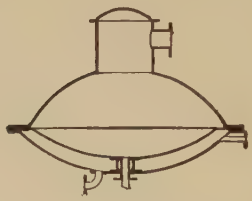


FIG. 6.—Howard vacuum pan.

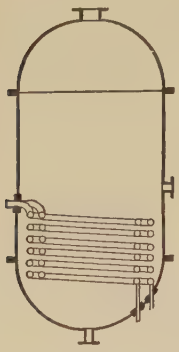


FIG. 7.—Immersed coil evaporator.

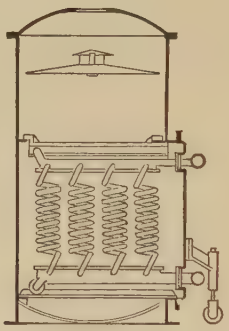


FIG. 8.—Reilly evaporator.

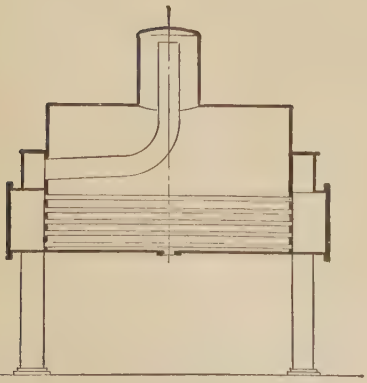


FIG. 9.—Rillieux horizontal tube evaporator.

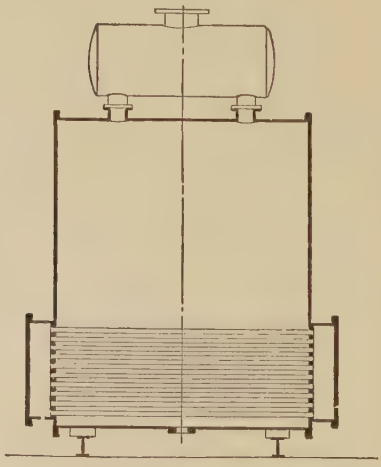


FIG. 10.—Yelinek evaporator.

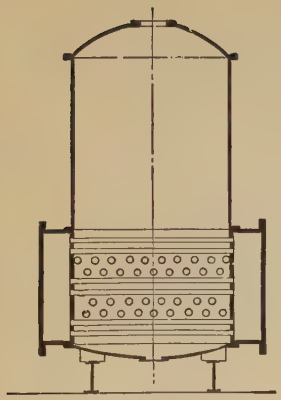


FIG. 11.—Horizontal tubes and vertical shell.

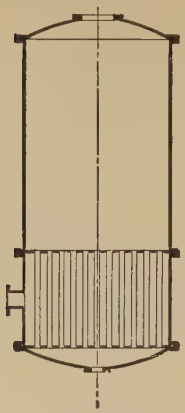


FIG. 12.—Vertical tubes and vertical shell.

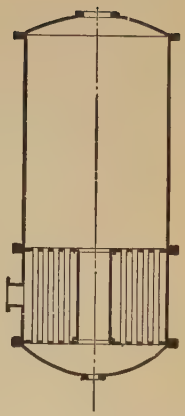


FIG. 13.—Claassen type.

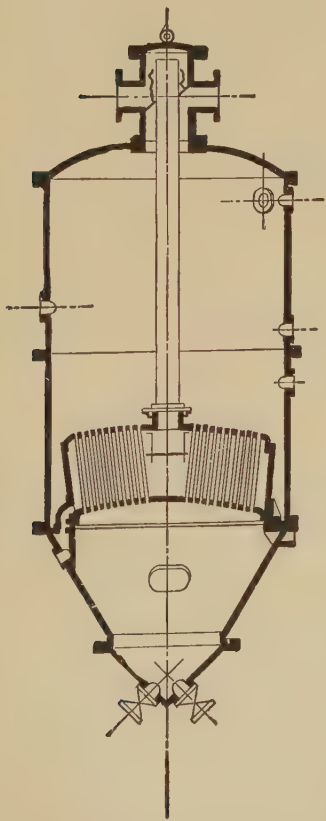


FIG. 14.—Annular downtake.

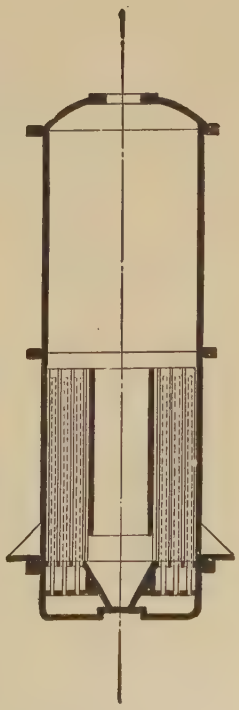


FIG. 15.—Steam inside, liquor outside tubes.

Figure 7; A considerable improvement as far as capacity is concerned is obtained by the use of tubular coils placed in the liquid to be evaporated. Large amounts of heating surface can be concentrated in a small space and this type of pan is still used to a large extent in the sugar and milk industry. Figure 8 shows the Reilly

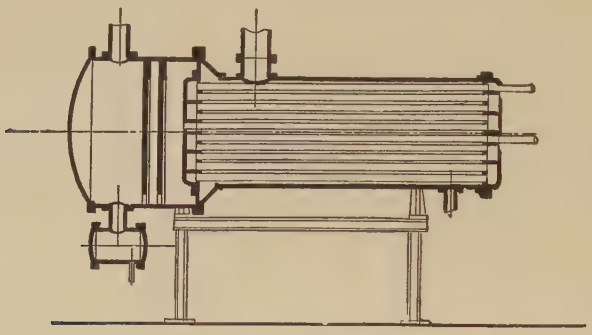


FIG. 16.—Yaryan high-speed circulation.

patent evaporator, a peculiar adaption of the coil type. This evaporator is mostly used for water stills as it is claimed by the inventor that on account of the vibration of the spiral coils, the scale will not deposit on the tubes and the heating surface can be kept clean easily.

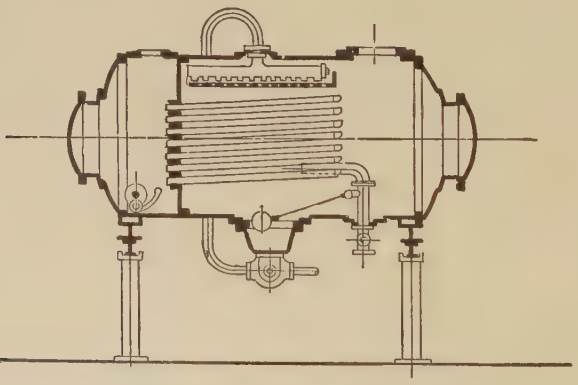


FIG. 17.—Lillie film evaporator.

Horizontal Type.—Figure 9 shows the first construction of an evaporator with horizontal tubes, invented and patented by Norbert Rillieux. The evaporator was used for the concentration of sugar juice in Louisiana, and is really the prototype of all modern horizontal-type evaporators. Figure 10: This construction was designed by Wellner Yelinek, and is being used to a large extent in the beet-sugar industry in this country and abroad. The same type, with a few minor changes, was patented by Swenson. Figure 11: A horizontal tube machine with a vertical cylindrical shell which was first designed by Simirenko, and later copied by Vis, and patented in the United States by Gibson and Zaremba.

Vertical Type.—Figure 12: The first vertical tube evaporator was built by Robert, and has the one great advantage over the horizontal type, that it can easily be cleaned with any ordinary flue cleaner. Figure 13: The same construction as the Robert evaporator with an addition of a very large downtake in the center to improve the circulation of the liquor. This type was first constructed by Claassen, and is now used to a large extent in the cane-sugar and malt-extract industries under the name of "Standard Evaporator."

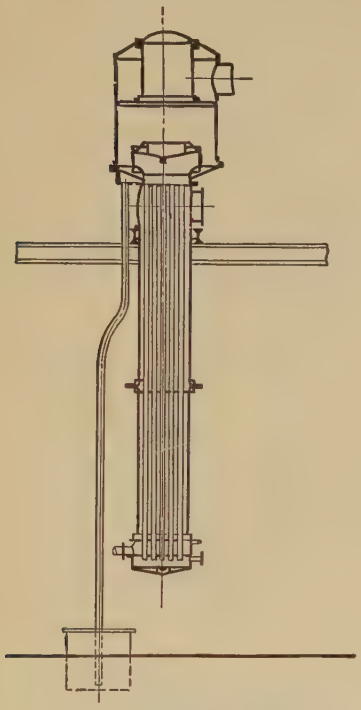


FIG. 18.—Original climbing-film evaporator (Kestner).

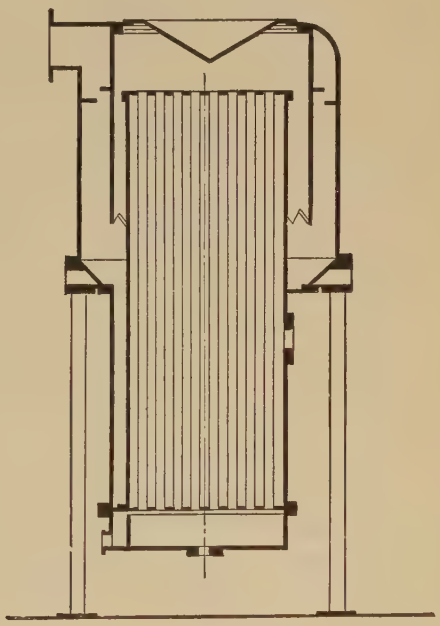


FIG. 19.—Mantius rapid-circulation evaporator.

Figure 14 shows an evaporator where the central downtake has been replaced by a large annular downtake. The steam chest does not form an integral part of the evaporator, but is placed inside the shell on separate supports. This construction was first built by Kauffmann, and introduced in the United States by Mantius. Figure 15: A vertical-tube evaporator with the steam inside and the liquor outside the tubes. The top of the tubes is closed, and the steam either enters through the main tubes (Sanborn), or through a small concentric tube (Pecqueur and Mantius).

Film Type.—Figure 16 shows the construction of Yaryan, in which the liquor is passed successively through a number of coils at very high speed. Figure 17: A special type of evaporator, Lillie patent, with horizontal tubes, where a thin film of liquor is formed on the outside of the tubes by the use of pumps. Figure 18 shows the first type of a climbing-film evaporator (Kestner). The liquor passes through the tubes only once, and is discharged from the vapor dome directly to the next evaporator or storage tank.

Figures 19 and 20 shows also a similar type of evaporator as constructed by Mantius and Badger, but in both cases the liquor is recirculated from the vapor body to the liquor chest, and is only discharged when it has reached the correct density.

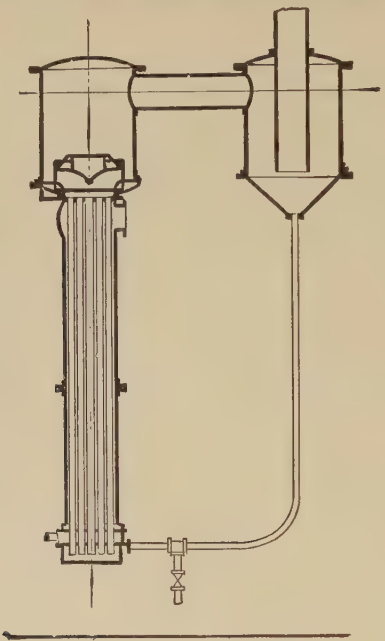


FIG. 20.—Badger rapid-circulation evaporator.

AUXILIARIES

Salt Filters.—A great many of the solutions to be handled in the chemical and allied industries separate salts during the concentration, either as a main product or by-product. In any case, it is necessary to separate this salt from the liquor, and this is accomplished by either discharging the mixture of liquor and salt into an open salt filter from where the liquor is returned to the evaporator, or sent to storage; or by separating the two products in a closed salt filter; or by discharging the mixture of the two into a trough and a rotary salt filter, where the salt will stick to the surface of the drum, and the solution is discharged by the pump. It is beyond the scope of this article to go into further details, and it is only to be mentioned that considerable progress has been made during the last few years in the handling of these salts, which ordinarily required considerable labor and expense.

Condensing System.—In the majority of cases, the vapors coming from the last effect must be condensed in some type of condenser, depending on the vacuum under which the plant is to be operated. The condenser may be of any standard design, but it should be borne in mind there are sometimes large fluctuations in the amount of vapor going to the condenser, and also that an evaporat-

NOTE.—A list of the most prominent manufacturers of these evaporators will be found in "The Chemical Catalog."

ing system will have considerable more air leakage than a standard engine or turbine. Therefore the condenser and vacuum pump should always be very much larger than is usually figured for standard steam practice.

Preheaters.—Under "Steam Consumption," it has been shown that the temperature of the incoming liquor has considerable influence on the steam consumption of the plant. Wherever possible, waste steam or waste gases should be used to preheat the cold liquors, so that they will enter the evaporator at about the boiling temperature in the first effect. For larger installations it is advisable to place preheaters in all the vapor lines of the multiple effect, and pass the liquor from one heater to the other, utilizing the vapors coming from each effect. This would mean that the liquor is preheated in multiple effect, and larger plants will show a considerable saving by this method of operation.

Separators and Catchalls.—To prevent losses by entrainment or foaming, it is necessary to install efficient separators and catchalls. For ordinary conditions, standard so-called "oil separators" with baffle plates may be used, but for extreme cases, separators of the centrifugal type, or very large expansion tanks with baffles must be installed. Prof. E. W. Kerr has given the results of some tests made with a Swartwout separator, which has shown extremely high efficiency (See *Bull.* 138 and 149 of the Louisiana State University).

EVAPORATORS IN THE CHEMICAL AND ALLIED INDUSTRIES

Specific Uses of Evaporators.—The following pages will give in convenient form the information and data that are necessary for the design or selection of a suitable evaporator for various kinds of solutions. The figures for capacity are given from practical experience under ordinary working conditions, and may be exceeded in many cases, especially in new plants where the heating surface and all joints are in first class condition.

INORGANIC PRODUCTS

Distilled Water.—Water will boil with a small temperature difference, and the apparatus is simple and cheap. It is therefore economical to combine a number of effects to one unit, and quadruple- and decituple-effect evaporators of the vertical-tube and film type are common practice. They are operated under pressure and vacuum, and the capacity will vary from 3 to 6 gal. per square foot according to the total temperature difference. Evaporator shells are made of cast iron or steel, and tubes of steel, brass or copper. There is no foaming, but generally all raw water contains considerable amounts of scale forming material and tubes have to be cleaned frequently. Sea water has to be used to a considerable extent for the manufacture of distilled water, and usually the Reilly coil type is used aboard ships, as it is claimed by the inventor that scale will not deposit on the tubes. Sometimes the 20°Bé. liquor coming from the sea water stills is concentrated further for the recovery of sea salt in standard vertical-tube evaporators with salt filters.

Ammonium Chloride.—The weak liquor will contain from 6 per cent to 10 per cent of salt, and is usually concentrated to 40 per cent in a horizontal-tube or vertical-tube evaporator. Capacity is about 2 gal. per square foot with a

steam pressure of 5 lb. and a vacuum of from 26 to 27 in. Salt crystals are recovered from the 40 per cent liquor by cooling, and in some cases they have been separated directly in a vertical-tube evaporator and recovered in a standard salt filter. In this case the capacity is from 1 to 1½ gal. per square foot. In some cases, evaporators have been made of cast iron with steel tubes, but other manufacturers are using evaporators lined with lead or acid-proof brick, and extra-heavy copper and bronze tubes.

Ammonium Nitrate.—The weak liquors are usually concentrated from 25 to 80 per cent in single effects of the horizontal-tube, or special vertical-tube evaporators. The steam pressure is from 5 to 20 lb. and the vacuum from 18 to 26 in. Evaporators have been built with cast-iron shells and cast-iron tubes, and also with aluminum tubes and enameled bodies.

Ammonium Sulphate.—Gas-producer plants will furnish weak liquors with about 30 per cent sulphate, and these liquors are concentrated in a vertical-tube evaporator until the salt crystals separate and drop into the salt filter attached to the evaporator. With a steam pressure of 5 lb. and a vacuum of 27 to 28 in., the capacity is from 1 to 1½ gal. per square foot. Evaporator shells are made of cast iron with tubes of bronze. Frequently the whole apparatus is lead-lined and the tubes are of lead, in which case the capacity will be reduced to about 60 per cent of the above figures.

Aluminum Sulphate.—The liquor usually contains free sulphuric acid and is concentrated from 10 to 45 per cent. It will solidify at 48 per cent. The evaporators must be made of lead entirely and the capacity will be ½ gal. per square foot with a steam pressure of 5 lb. and a vacuum of 26 in. Vertical-tube and horizontal-tube machines may be used.

Barium Chloride.—The usual concentration is from 10 to 50 per cent, and with a horizontal-tube evaporator the capacity will be 1 gal. per square foot, with a steam pressure of 5 lb. and a vacuum of 26 in. Number of effects depends on quantity of liquor. Evaporators are built of cast iron or steel, with steel or wrought-iron tubes.

Calcium Chloride.—The weak liquors will contain from 7 to 10 per cent of solids, and are concentrated to from 43 to 48 per cent. Sodium chloride is separated and recovered in standard salt filters. For the preconcentration, a vertical-tube and rapid-circulation type should be used; for the final concentration, a vertical-tube machine only. The capacity is 2 gal. per square foot for the weak liquor, and 1 gal. for the final concentration, assuming a steam pressure of 10 lb. and a vacuum of 28 in. The boiling point of calcium chloride increases considerably with the concentration, and the liquors usually separate a heavy scale of calcium sulphate, which has to be removed by mechanical cleaning.

Calcium and Magnesium Chloride (Mother liquors from salt works).—Weak liquor 29°Bé., heavy liquor 45°Bé. The concentration is done in horizontal-tube and vertical-tube single effects at a capacity of 1 gal. per square foot, with a steam pressure of 10 lb. and a vacuum of 26 in. Sodium chloride will separate. Final concentration to 59°Bé. is done in open pans with high-pressure steam. All parts must be of cast iron or wrought iron, as copper will discolor the product.

Magnesium Sulphate (Epsom salt).—Concentrated from 48 per cent or less to 75 per cent in vertical-tube or rapid-circulation type evaporators. With a

steam pressure of 15 lb. and a vacuum of 20 in., the capacity is from 1 to 1½ gal. per square foot. Single effects for small quantities; double effects operated counter-current for larger volumes. Evaporators are made of cast iron with wrought-iron or copper tubes. Natural epsom-salt solutions will deposit a scale of sodium sulphate on the tubes, and the solutions produced from dolomite and sulphuric acid will form a heavy scale of calcium sulphate. Tubes must therefore be accessible for mechanical cleaning.

Potassium Bichromate.—Concentrated from 25 to 50 per cent in a vertical-tube evaporator with a steam pressure of 5 lb. and a vacuum of 26 in., at a rate of 1 gal. per square foot. Sodium chloride is separated in salt filters. Evaporators must be built of cast-iron shells and charcoal-iron or steel tubes.

Potassium Carbonate.—Density of weak liquor will vary greatly, dependins on the source of the product. Solutions are concentrated until the salt crystal. separate in a vertical-tube evaporator with a steam pressure of from 5 to 10 lb and a vacuum of 26 in., at a capacity of 1 gal. per square foot. The carbonate crystals are recovered in salt filters. Evaporators are made of cast iron or steel, and tubes of steel or charcoal iron.

Potassium Chloride.—Density of weak liquor solution will depend on the source of material. It will vary from 5 to 20 per cent of solids. It is usually concentrated up to the crystallization point, and crystals are recovered by cooling in crystallizing vats. Capacity will be from 1 to 1½ gal. per square foot, with a steam pressure of 5 lb. and a vacuum of 26 in. Horizontal-tube or rapid-circulation type evaporators may be used, and vertical-tube evaporators must be used for liquors that also separate sodium chloride. Cast-iron shells with either charcoal-iron or copper tubes are used.

Potassium Hydroxide.—Usually concentrated from 10 to 46 per cent in multiple-effect evaporators of vertical-tube construction, with a steam pressure of from 5 to 20 lb. and a vacuum of from 26 to 28 in., at a capacity of about 1 gal. per square foot. Potassium carbonate or chloride are recovered in salt filters. Evaporator shells are of cast iron and tubes of charcoal iron.

Potassium Nitrate.—Concentrated from 30 to 70 per cent in a vertical-tube evaporator with a capacity of 1 gal. per square foot, and a steam pressure of 15 lb. and a vacuum of 27 in. A double effect operating counter-current is advisable, and the sodium chloride is recovered in salt filters. Evaporator bodies are of cast iron and tubes of copper or charcoal iron.

Potassium Sulphate.—The weak liquors containing from 10 to 20 per cent of sulphate crystals are concentrated in vertical-tube evaporators with a steam pressure of about 5 lb. and a vacuum of 27 to 28 in., at a capacity of 1 to 1½ gal. per square foot. The sulphate crystals are recovered in salt filters attached to the evaporators, which are built of either cast-iron or steel with steel or charcoal-iron tubes.

Sodium Bichromate.—Liquors containing from 50 per cent bichromate are concentrated for the separation of the sodium chloride in vertical-tube evaporators, with a steam pressure of 5 lb. and a vacuum of 26 in. at a capacity of 0.8 gal. per square foot. The sodium chloride is recovered in salt filters. Multiple effects should be operated counter-current, and evaporators can be built of cast iron or steel with steel or charcoal-iron tubes.

Sodium Carbonate.—The weak solutions containing about 10 per cent of

crystals are concentrated in vertical-tube evaporators with exhaust steam at 5 lb. and a vacuum of 26 in., at the rate of 1¼ gal. per square foot. Sodium carbonate is separated in salt filters and a rapid circulation between salt filter and evaporator must be maintained in order to prevent crystallization of the carbonate on the tubes. Evaporators are built of cast iron with steel or charcoal-iron tubes.

Sodium Chloride (Common salt).—One ton of salt will require the evaporation of about 3 tons of water from a saturated brine. All fine-grained salt (vacuum salt) is made in single- or multiple-effect evaporators of the Standard type with deep cone bottoms, which will give a capacity of from 1¼ to 1⅓ gal. per square foot, with a steam pressure of 5 lb. and a vacuum of 28 in. Usually these evaporators have large central downtakes, and in some cases, special propellers for better circulation of the brine. Very large units have been built by Manistee, having a capacity of from 300 to 500 tons of salt per day. Salt is recovered in closed filters, by continuous vacuum filters, or with bucket elevators. Evaporation is retarded by incrustations of salt, and very frequently by scaling of the tubes. Without proper treatment of the brine, this scale has to be removed mechanically once a day. Some of the brines will foam considerably, and a high-vapor space is necessary. Evaporators are usually built with cast-iron shells and copper or brass tubes. Sometimes iron tubes are also used.

Coarse-grained salt, so-called "grainer salt," is manufactured in large shallow pans made of steel or concrete, and heated by coils. The salt is removed continuously by scrapers which drag along the bottom of the pan. With low-pressure steam these grainers have a capacity of from ⅓ to ½ gal. of water evaporated per square foot of heating surface.

Sodium Chlorate.—Liquors are concentrated from about 40 to 56 per cent, and sodium chloride is separated. The work is usually done in horizontal-tube or special vertical-tube double-effect evaporators in counter-current, with a steam pressure of 5 lb. and a vacuum of 26 in., at a rate of 0.8 gal. per square foot. Evaporator shells and tubes should be of cast iron.

Sodium Hydroxide (Caustic soda).—Liquors are usually concentrated from 10 to 46 per cent in vertical-tube multiple-effect evaporators with a steam pressure of 20 lb. and a vacuum of 27 to 28 in., at the rate of from 1 to 1¼ gal. per square foot. Sodium carbonate and chloride are separated during the evaporating process, and are recovered in salt filters. Evaporators are usually built of cast iron or steel, with steel or charcoal-iron tubes. For the more concentrated liquors copper tubes have also been used successfully.

The 46 per cent solution is further concentrated to 80 per cent in the Mantius high concentrator with a steam pressure of 100 lb. and a vacuum of 27 in. at the rate of 0.6 gal. per square foot. Evaporator of cast iron, tubes of special cast iron or copper.

Sodium Nitrate.—Solutions are concentrated from 30 to 70 per cent in a vertical-tube evaporator with a steam pressure of 5 lb. and a vacuum of 27 in., at the rate of 0.9 gal. per square foot. For larger quantities a double or triple effect operating counter-current is used. Evaporators are built of cast iron, tubes of charcoal iron or copper. Sodium chloride is separated during the first stage of evaporation and sodium nitrate during the final concentration.

Sodium Nitrite.—Liquors are concentrated from 30 to 80 per cent in a horizontal-tube machine with a steam pressure of 5 lb. and a vacuum of 26 in.,

at the rate of 0.7 gal. per square foot. Quantities are usually small and only single effects are used. Evaporators of cast iron, tubes of aluminum.

Sodium Phosphate.—Solutions are concentrated from 6 to 24 per cent in a horizontal-tube double-effect evaporator with a steam pressure of 5 lb. and a vacuum of 26 in., at the rate of 1½ gal. per square foot. Evaporators are of cast iron or steel, with tubes of steel.

Sodium Silicate.—Concentrated from 50 to 60 per cent in a horizontal-tube single effect with a steam pressure of 5 lb. and a vacuum of 26 in., at the rate of ⅓ to ½ gal. per square foot. Evaporators are built of cast iron with steel tubes. Capacity per square foot of heating surface is very low on account of the heavy coating of the tubes.

Sodium Sulphate.—The saturated solution of from 30 to 33 per cent solids is concentrated in single or multiple-effect evaporators of the vertical-tube type, with a steam pressure of 5 lb. and a vacuum of 26 in., at the rate of from 1 to 1¼ gal. per square foot. The sodium sulphate is recovered in salt filters, and a rapid circulation between evaporator and salt filters must be maintained in order to prevent heavy coating on the tubes.

Sodium Sulphide.—Concentrated from 8 to 45 per cent in single or multiple-effect evaporators of the horizontal-tube or rapid-circulation type, with a steam pressure of 5 lb. and a vacuum of 26 in., at the rate of 1½ to 2 gal. per square foot. Evaporators are of cast iron with tubes of charcoal iron or steel. Sometimes the liquor is further concentrated to 60 per cent in the Mantius high concentrator, which is equipped with cast-iron tubes.

Sulphuric Acid.—Weak acid solutions are successfully concentrated in multiple-effect evaporators of the vertical-tube or horizontal-tube type, having cast-iron shells and lead lining, with lead tubes; in some cases the whole apparatus is built of lead. Capacity, steam pressure and vacuum depend on special conditions.

ORGANIC PRODUCTS

Sugar, Glucose and Maltose.—One ton of sugar cane will give about 150 lb. of sugar in Louisiana, and from 200 to 225 lb. in the tropics. The juice will amount to 200 gal. per ton, with a density of from 5½ to 8°Brix. It is concentrated in a triple or quadruple-effect evaporator to about 27°Bé., with a steam pressure of 10 lb. and a vacuum of from 27 to 28 in. The capacity is 3 to 4 gal. per square foot, and the Standard, Lillie, Kestner, Sanborn and Swenson evaporators are the usual types. The 27°Bé. solution is concentrated to 42° in single effects of the coil or Standard type. The capacity is about 1 gal. of water evaporated per square foot, and either live or exhaust steam is used in these pans.

Beet Sugar.—One ton of beets will give about 1¼ ton of juice, having a density of from 7 to 8½°Bé. The concentration is done in two stages, and triple- or quadruple-effect evaporators with a capacity of from 3 to 4 gal. per square foot are used for the concentration up to 27°Bé. The Wellner Yelinek, Swenson and Kilby construction are the usual types. The final concentration up to 42°Bé. is done in single effects of the coil or Standard type, with a capacity of about 1 gal. per square foot.

A properly purified and well-filtered solution will not give much trouble on account of foaming, but all sugar juices contain considerable amounts of lime and silica, which will form a hard scale on the tubes. This scale is usually removed by boiling with a weak soda ash and acid solution. The Standard type has the advantage over all other constructions as it can be cleaned mechanically and is therefore used to a large extent in the cane-sugar industry. Evaporator shells are usually made of cast iron and sometimes of steel. Tubes are of brass or copper.

Glucose.—One bushel of corn will give about 14 gal. of liquor, and the first concentration is done in a triple-effect evaporator from 15 to 30°Bé. with from 5 to 10 lb. of exhaust steam pressure and a vacuum of from 27 to 28 in. The capacity is from 2½ to 3 gal. per square foot. Usually the standard and the horizontal-type evaporators are used; also the semi-film. The latter has the advantage that it will prevent foaming and facilitate cleaning. The second concentration from 30 to from 44 to 45°Bé. is done in a single-effect of the standard or horizontal type. The capacity is from ¾ to 1 gal. per square foot, and the horizontal type must be of special construction so as to leave sufficient space between the tubes for the discharge of the heavy syrup. On account of the calcium sulphate in the weak liquor, the tubes in the multiple effect will foul very rapidly, and have to be cleaned either chemically or mechanically. The liquors are apt to foam, and special precautions in the form of extra vapor space and separators are to be taken. Evaporators are usually built with cast-iron shells and copper tubes.

Steepwater.—One bushel of corn will give from 4 to 5 gal. of steepwater and this is concentrated in vertical or horizontal-type evaporators from 2½ to 23°Bé., and sometimes 30°Bé. Usually single effects only are used, as the higher temperatures of multiple effect would give the liquor a dark color. A concentration over 23°Bé. will also darken the liquor. The capacity is from 1½ to 2 gal. per square foot, but tubes foul quickly on account of a coating of gluten, which has to be dissolved by strong alkaline solutions. The liquor is always slightly acid, and only cast-iron shells with copper tubes can be used. High-vapor space and special entrainment separators are necessary to avoid losses by foaming.

Maltose.—Malt extract or baker's extract with a high percentage of diastase are made from pure malt, and one ton of malt will give about 4 tons of 14°Balling, and 6 tons of 10°Balling weak extract. This liquor is usually concentrated to 50°Balling in a single or double effect of the horizontal or semi-film type, with a capacity of from 2½ to 3½ gal. per square foot, with atmospheric pressure or less in the steam chest and 28 in. vacuum. For all diastatic extracts, the boiling temperature must not exceed 60°C. The 50°Balling extract is concentrated to from 75 to 80°. in a single-effect of the standard or horizontal type, with a capacity of about ½ gal. per square foot.

Maltose syrup is made from corn flour and malt, and 1 ton of flour with 0.2 tons of distillers malt will give about 8 tons of 10°Balling extract. After proper filtration, this liquor is then concentrated to 50° in a double or triple effect of the horizontal or semi-film type, with a capacity of from 2½ to 3½ gal. per square foot. The 50° extract is concentrated to 80° in a single effect of the vertical or horizontal type.

All the extracts will foam considerably, and special precautions should be taken to prevent serious losses. The semi-film type has been used very successfully. Unfiltered extracts will produce considerable scale on the tubes, consisting mostly of albuminous matter, and this scale has to be removed frequently by either boiling with

chemicals or mechanical cleaning. The horizontal-type machine is not very suitable as the tubes cannot be cleaned mechanically.

Evaporators are usually built with cast-iron shells and copper tubes. In some cases evaporators have been built of copper entirely, but experience has shown that this is not necessary in order to make a high-quality extract.

Milk.—For the manufacture of condensed or evaporated milk, skimmed milk is usually concentrated to ⅓ or ¼ of its original volume, and this is done in single-effect evaporators of the film or coil type. The capacity is from 2 to 3 gal. per square foot, with low-pressure steam, and up to 4 gal. per square foot with high-pressure steam, at a vacuum of from 26 to 28 in. So as not to cook the milk, the temperature must not exceed 60°C., and in Europe skimmed milk has been concentrated in double-effect evaporators of the rapid-circulation type with good success.

Milk-whey is concentrated to recover the milk sugar after the albuminous matter has been removed by acidification. The filtered liquor is concentrated from 2½ to 20°Bé. in double-effect evaporators of the horizontal or rapid-circulation type, with a capacity of from 2½ to 3 gal. per square foot, with exhaust steam and from 26 to 28 in. vacuum. This solution is further concentrated to about 40°Bé. in a single effect of the coil type, having a capacity of about ¾ gal. per square foot. In all cases special precautions must be taken to avoid losses by foaming, and the rapid-circulation type has been very successful on this account. Milk-whey will cause a scaling of the tubes and frequent cleaning is necessary. Evaporators for condensed or evaporated milk should be built of copper, and all parts must be accessible for frequent cleaning and scouring. Milk-whey evaporators are built with cast-iron shells and copper tubes.

Glue.—Hide-glue is usually concentrated from 2 to 15 per cent solids, and bone glue from 3 to 20 per cent in a horizontal-tube or rapid-circulation type double or triple effect evaporator. The capacity is from 2 to 2½ gal. per square foot with a steam pressure of 5 lb. and a vacuum of 27 in. Evaporators are built of cast iron with copper or brass tubes.

Gelatin.—Methods of manufacture vary a great deal, but usually the concentration is from 4 to 10 per cent, and the capacity of the evaporators may be figured as 3 gal. per square foot with a steam pressure of 0 lb. and a vacuum of 27 to 28 in. Only single effects or double effects may be used, as higher temperatures will spoil the product. Ordinary gelatin will require evaporators with cast-iron shells and copper tubes, the same as for glue. High-grade gelatin should be made in machines built of copper, monel metal or nickel.

Beef Extract.—Depending on the nature of the extract to be made, the points of concentration will vary and may be assumed from 2 to 10 per cent, and also from 4 to about 60 per cent. The capacity will vary from 2 to 3 gal. per square foot with 5 lb. steam pressure and from 26 to 27 in. of vacuum. The horizontal-type machine is used for this purpose and is built with cast-iron shell and copper tubes.

Tanning Extract.—The strength of the weak liquors varies with the raw material. The leaching of chestnut chips will give extracts containing from 3 to 4 per cent solids, while Quebracho and Campeche chips will give liquors containing from 7 to 10 per cent of solids. They are usually concentrated to about 50 per cent in a multiple-effect evaporator, with a capacity of from 3 to 4 gal. per square foot and a steam pressure of 5 lb. and a vacuum of 27 to 28 in. The 50 per cent extract is sometimes concentrated further in special single-effect

evaporators to 75 or 80 per cent. In order to prevent contamination of the extracts and corrosion, it is necessary to make these evaporators of copper and bronze entirely. The weak liquors will foam considerably and scale will form on the tubes especially if hard water is used for the leaching process. The vertical-tube evaporator is commonly used for tanning extracts, but the rapid circulation and film types are excellent machines for this purpose.

Cascara, Coffee, Licorice, Nicotine, Tea.—The original density of these extracts depends entirely on the manufacturing process, and no average can be given. The liquors are usually concentrated to from 40 to 50 per cent solids, and the capacity is about 2 gal. per square foot with a steam pressure of 5 lb., and a vacuum of from 27 to 28 in. Single effects of the horizontal-tube or rapid-circulation type are standard equipment; sometimes a plain coil pan with or without steam jacket is used. Heating surface must be of copper and the shells can be of either copper or enameled steel.

Tartaric Acid.—The usual concentration of the filtered solution is from 30 to 70 per cent of solids, and the work is done in single-effect evaporators made entirely of hard lead with extra-heavy lead coils. The all-lead construction may be replaced by a cast-iron shell with suitable lead lining, and the tubular heating surface with vertical tubes and lead flueplates can be used instead of the lead coils. It is very important that the construction is such that tubes can readily be cleaned from the hard scale produced by the calcium sulphate in the solution. With a steam pressure of from 30 to 50 lb. in the coils, and a vacuum of 27 in., the capacity is from 1½ to 2 gal. per square foot.

Lactic Acid.—The filtered solution is usually concentrated from 8 to 50 per cent in a single-effect evaporator of the vertical-tube or rapid-circulation type. All parts of the evaporator coming in contact with the liquor or vapor must be of copper. The solution contains calcium sulphate, and the deposit of hard scale must be removed very frequently from the tubes.

Pyroligneous Acid.—One cord of wood will give from 250 to 300 gal. of crude acid. A mixture of acetic acid, alcohol and water is distilled from the crude acid in double- or triple-effect evaporators of the vertical-tube or rapid-circulation type. The total evaporation is from 90 to 95 per cent, and the capacity is from 2 to 3 gal. per square foot, with a steam pressure of 5 lb. and a vacuum of 27 in. A surface condenser is attached for the recovery of the watery acid and alcohol. Evaporators must be built entirely of copper. Heating surface is frequently covered by a heavy coating of tar and charcoal dust, which has to be removed by mechanical cleaning or may be dissolved by the crude acid.

Acetate of Lime.—One cord of wood will give from 180 to 220 gal. of solution which is usually concentrated from about 6 to 28 per cent and sometimes as high as 30 per cent solids. With a steam pressure of 5 lb., and a vacuum of 27 in., the capacity is from 1½ to 2 gal. per square foot in double or triple effects of the vertical-tube or rapid-circulation type. Evaporator shells are made of cast iron with copper or steel tubes.

For smaller installations, the concentration of the crude pyroligneous acid and acetate of lime is done in a double-effect evaporator, with the acid in the second effect.

Acetate of Soda.—The usual concentration is from 5 to 33 per cent and from 33 per cent to the crystallizing point, and the salts are recovered in standard salt

filters. For the pre-concentration, the capacity is from 1½ to 2 gal. per square foot in a horizontal-tube evaporator, and for the crystallization a vertical-tube machine is used with a capacity of about 1 gal. per square foot. The steam pressure is 5 lb., and the vacuum 27 in. Evaporators are built with cast-iron shells and steel or wrought-iron tubes.

Sodium Benzol Sulphonate.—These solutions are concentrated from 10 to 50 per cent solids in double or triple effects of the horizontal-tube, vertical-tube, or rapid-circulation type, with a capacity of from 2 to 2½ gal. per square foot, at a steam pressure of 5 lb. and a vacuum of 27 in. Cast-iron or steel shells with wrought-iron or steel tubes is common practice.

Glycerin.—In the soap industry, 1,000 lb. of fat will give about 2,000 lb. of soap lyes containing 5 per cent glycerin and about 10 per cent of salts. These liquors are usually concentrated in a double- or triple-effect evaporator of the horizontal-tube or vertical-tube type to from 40 to 50 per cent of glycerin. With a steam pressure of 5 lb. and a vacuum of 27 in., the capacity is about 1 gal. per square foot. The 50 per cent solution is further concentrated to about 80 per cent in a single effect.

In the candle and oil industry, large quantities of glycerin are produced by the Twitchell process. The liquors are usually concentrated from 15 to 80 per cent in single-effect or double-effect evaporators of the vertical-tube, horizontal-tube or rapid-circulation type. The capacity is from 1½ to 2 gal. per square foot with a steam pressure of 5 lb. and a vacuum of 28 in. All glycerin solutions should be handled in evaporators with special high-vapor space and large catchalls or entrainment separators to prevent losses by foaming. Glycerin produced by the Twitchell process will form a hard scale on the tubes, consisting mostly of calcium sulphate.

Grape Juice and Cider.—The initial density of these liquors will vary from 6 to 8°Bé., and the final concentration is usually carried to from 30 to 33°Bé. The capacity is from 1½ to 2 gal. per square foot with a steam pressure of 5 lb. and a vacuum of from 26 to 28 in. Only single effects can be used in order to avoid high temperature and a cooked taste in the liquor. The evaporators must be built of copper, which is sometimes tinned. Machines of the vertical-tube or rapid-circulation type are commonly used, but a number of factories are also using the ordinary coil pan with shells made of copper or enamel steel.

RECOVERY OF WASTE LIQUORS

Black Liquor from Soda-pulp Mills.—One ton of pulp will produce about 3,300 gal. of 5 deg. liquor, and this is concentrated to 35 deg. in multiple-effect evaporators of the horizontal and film type, with a capacity of from 2 to 2½ gal. per square foot, according to the steam pressure, which will be from 10 to 25 lb., with a vacuum of from 27 to 28 in. Special construction and separators are necessary on account of the excessive foaming. Evaporators must have cast-iron or steel shells with wrought-iron or steel tubes, as the liquors are strongly alkaline.

Sulphate Liquor.—One ton of sulphate pulp which is used for the manufacture of Kraft paper produces about 2,000 gal. of liquor which is to be concentrated from 10 to 20°Bé. Usually a double or triple effect of the horizontal type is used for this purpose, and the capacity is from 2 to 2½ gal. per square foot according to the steam pressure. The concentration of this 20°Bé. liquor is continued

and finished in a Carlson evaporator, utilizing the waste heat from the rotary furnaces. Evaporators must have cast-iron or steel shells and wrought-iron tubes. Precautions against foaming are very necessary.

Evaporators for the recovery of soda or sulphate pulp liquors must be washed frequently in order to remove the scale which is formed by the organic matter in the liquor.

Sulphite Waste Liquor.—One ton of pulp will produce from 2,500 to 5,000 gal. of waste liquors, the density of which will vary from 6 to 10 per cent. It is usually concentrated after neutralization to about 30°Bé. (50 per cent solids) in multiple-effect evaporators of the vertical or rapid-circulation type, and the capacity is from 2½ to 3½ gal. per square foot, with a steam pressure of 5 lb. and a vacuum of from 27 to 28 in. The tubes will foul very quickly, on account of the large amount of calcium sulphate contained in the weak solution, and therefore evaporators are to be constructed so that tubes can readily be cleaned mechanically.

Evaporators are usually built with cast-iron or steel shells and with wrought-iron tubes. Sometimes, where the liquor is not quite neutralized, copper tubes are to be preferred. In special cases where the waste liquor is not neutralized at all, all copper construction is necessary, which may be replaced by evaporator shells of cast-iron or steel with acid-proof brick lining and copper tubes.

Mercerizing Liquors.—Quantities of liquor will vary considerably with the process of washing used in each plant, and a reasonable average will give 2 gal. of 6°Twaddell waste liquor for each pound of cotton mercerized. These liquors are usually concentrated to from 70 to 80°Twaddell in a triple- or quadruple-effect evaporator of the horizontal-tube or rapid-circulation type. The capacity is from 2½ to 3½ gal. per square foot, with a steam pressure of from 10 to 15 lb. Special construction and entrainment separators are necessary to prevent losses by foaming. The last effect will frequently show incrustations of sodium carbonate which can be dissolved in hot water or weak liquor. Evaporators are usually built of steel with steel or charcoal-iron tubes.

Wool-scouring Waste.—According to the method of treatment, the quantities of waste liquor recovered will vary quite a great deal, and 1,000 lb. of wool will give from 1,000 to 1,500 lb. of waste liquors containing about 5 per cent solids. They are concentrated to 60 per cent in a double- or triple-effect evaporator of the horizontal-tube or rapid-circulation type, and the capacity is from 2 to 3 gal. per square foot with a steam pressure of 5 lb. and a vacuum of 26 to 27 in. In some cases, the liquor is further concentrated to about 75 per cent with live steam pressure of from 30 to 50 lb., and at a capacity of about ½ gal. per square foot. The liquors are usually difficult to handle on account of excessive foaming, and tubes are apt to foul on account of incrustations consisting mostly of organic matter. Evaporators are built of steel with wrought-iron or steel tubes, as the liquors are strongly alkaline.

Distillery Slop.—One bushel of corn or rye will give from 25 to 30 gal. of liquor containing from 1½ to 2 per cent solids. They are concentrated to about 35 per cent solids usually in countercurrent, and the capacity is from 2¾ to 3 gal. per square foot with a steam pressure of 5 lb. and a vacuum of 27 in. Usually a triple- or quadruple-effect is used and good results have been obtained by reversing the current of the liquor through the evaporators; otherwise tubes will foul

very readily on account of incrustations of nitrogenous matter in the liquor. Distillery slop coming from cane or beet molasses is concentrated from 10 to 50° Brix in a triple- or quadruple-effect evaporator and from 50 to 80° in single effect. Sometimes the concentration is finished up to 80° in the multiple effect direct, with a steam pressure of 15 lb. and a vacuum of from 27 to 28 in. Evaporator shells must be of cast-iron with tubes of copper or brass and usually the horizontal-tube or rapid-circulation type are used.

Iron Sulphate (Copperas).—This material is recovered from pickling liquors in sheet and tube mills, and will contain about 40 per cent $FeSO_4.7H_2O$. The liquors are concentrated from 1.24 to 1.40 specific gravity in a double- or triple-effect evaporator with a capacity of from 2 to 3 gal. per square foot, and a steam pressure of 5 lb. and a vacuum of 27 in. The horizontal-tube machine has been used successfully for this purpose. The concentrated liquors are then crystallized by cooling. Another method of recovery is the concentration up to 1.45 specific gravity when the crystallization begins in the evaporator and the product has then to be handled in a vertical-tube evaporator with salt filters attached. The capacity is from 1½ to 2 gal. per square foot with a steam pressure of 5 lb. and a vacuum of 27 in. The liquors are usually slightly acid, and evaporator shells must be made of cast iron with the tubes of copper.

Tank Water from Packing Houses.—Sheep and calves will produce from 1½ to 2 gal., hogs from 2½ to 3 gal. and beeves from 7 to 10 gal. of tank water, with a density of about 4°Bé., and the quantity will be larger if the percentage of solids is lower. In most cases, 3°Bé. liquor is concentrated to about 27° in multiple-effect evaporators of the horizontal type having a capacity of 1.8 gal. per square foot, with a steam pressure of from 3 to 5 lb. and a vacuum of from 27 to 28 in. There is not much tendency to foam, but sometimes tank water will form a heavy scale on the tubes which has to be removed by boiling with alkaline solutions. Waste waters are sometimes slightly acid, and evaporators must have cast-iron shells and copper or brass tubes.

Tank Water from Garbage Plants.—One ton of garbage will produce from 280 to 300 gal. of waste waters with from 6 to 8 per cent of solids. These liquors are concentrated to 50 per cent in a multiple-effect evaporator of the horizontal type, with 5 lb. pressure and a vacuum of from 26 to 27 in.

Liquors will foam slightly, and cause fouling of the tubes on account of the organic matter settling on the heating surface. Evaporators are usually built of cast iron with copper or brass tubes, and the iron surface is protected from the liquor by an acid-proof brick lining. Garbage tank water usually contains a certain percentage of acetic and butyric acid.

Tank Water from Fisheries.—One ton of fish will give from 1,500 to 2,500 lb. of tank water, having a density of from 4 to 5°Bé. This liquor is usually concentrated to about 23°Bé. in multiple-effect evaporators of the horizontal type, having a capacity of from 1⅓ to 1½ gal. per square foot, with a steam pressure of 5 lb. and a vacuum of 26 in.; the liquor acts practically the same as packing-house tank water, and the same type of evaporator is used for this purpose.

Zinc Chloride.—This material is recovered by systematic washing of hard fibre. The diluted liquor has a density of from 16 to 20°Bé., and is concentrated to from 68 to 72°Bé. in a double-effect evaporator which has a capacity of from 1.3 to 1.8 gal. per square foot, with a steam pressure of from 5 to 20 lb. and a vacuum of 27

in. The boiling point of the liquor goes up very rapidly during the final concentration. The liquors become acid during evaporation and therefore all parts of the evaporators coming in contact with the liquor must be made of acid-proof bronze with extra-heavy copper tubes. Vapor belts can be made of cast iron.

DRYING

Bibliography and History.—In spite of the fact that the art of Drying is a very important branch of the chemical industry, there was up to a few years ago not a single textbook giving specific information and data on this subject. Only comparatively recently three books have appeared: E. Hausbrand, "Drying by Means of Air and Steam," Thos. C. Marlow, "Drying Machinery and Practice" (with a chapter, 'Bibliography of Drying and Dessicating'); and Otto Marr, "Das Trocknen and Die Trockner" (Drying and Drying Machinery). Marr's book is by far the most complete treatise on this subject, and it is to be regretted that it has not been translated into the English language, as it contains valuable data regarding the theory and practice of drying.

The art of Drying must naturally be a very old one, but the author has been unable to find any accurate information as to the invention and first application of the various types of dryers. Vacuum dryers were introduced only about 35 years ago by Passburg.

Theory of Drying.—Drying is the process of lowering the weight of a substance by reducing the amount of water or solvent contained in this substance. This may be accomplished by: Pressure, suction or decantation; evaporation, by means of hot air or gases, steam, hot water, oil and electricity. The application of pressure, suction or decantation can only remove a certain amount of moisture, and in order to make an article bone dry, it is always necessary to resort to heat and evaporate the water or solvent. Filter presses, centrifugals, hydraulic presses, stationary and revolving suction filters have been treated in other parts of this book, and this chapter will only deal with drying by evaporation.

In order to reduce operating expenses, the excess of water should always be removed by pressure, suction or decantation as far as possible, because drying by evaporation is rather expensive, and requires costly machinery. For instance, table salt and granulated sugar are first handled in centrifugals and then completely dried in rotary dryers with hot air.

Drying by evaporation requires heat for three purposes: first, to raise the temperature of the product to the required level; second, to evaporate the water or solvent; third, to replace the losses by radiation which in most drying processes form a large percentage of the total heat required. In some cases, it is also necessary to include the heat required for the heating of the shelves, racks and wagons carrying the material. The heat may be applied directly by air and gases, or indirectly by steam, hot water, oil and electricity.

Drying by Air and Gases.—Capacity and efficiency of an air-heated dryer always depends on atmospheric conditions, and such apparatus should therefore be figured for extreme conditions: in winter with cold, dry air, and in summer with warm, almost saturated air. Weather conditions interfere often very

seriously with the operation of a hot-air dryer, and for this reason dryers have been designed that make use of the same air over and over again by letting the excess vapor escape into the atmosphere or condense the moisture by cooling the mixture of air and vapors.

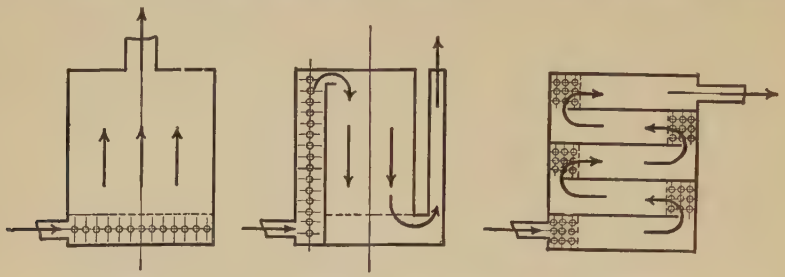
The air is usually heated by exhaust or live steam, and as explained in the previous pages on evaporation, the heat transmission depends on: (1) The temperature difference between air and steam; (2) the velocity of the air; (3) the amount of heating surface; (4) the condition of the heating surface, *i.e.*, the material and the diameter of the pipes or heater sections. Small pipes have a higher capacity than larger ones. According to Marlow, page 84, the greatest economical velocity of the air through the heater and dryer is about 20 ft. per second, and with air entering at 60°F., and steam at 220°, the heat transmission, K_o, will vary from 1.4 to 1.8 B.t.u. per square foot per hour for 1° temperature difference. Under these conditions of 160° temperature difference, cast-iron radiators will absorb about 260 B.t.u.; 2 to 4 in..wrought-iron pipes, 300 B.t.u.; and 1 to 2 in. pipes, 350 B.t.u. Carrier (*Trans. A. S. M. E.*, Vol. 33) gives for higher steam pressure (50 lb.) $K_o = 1/(a + (b/w))$, in which a and b are constants ($a = 0.045$ and $b = 55$ average), and w is the velocity of air in feet per minute. This equation gives much higher values for K_o than stated by Marlow. Practical data on this subject are given in the catalogs of heater manufacturers.

The exact determination of the amount of air and gases needed for the drying of a certain substance requires rather intricate calculations. In practically all cases, one must consider constants and factors that are determined by experience and actual tests only. However, in all dryer installations of this kind, the following points are important: (1) The quantity of hot air or gases required to raise the temperature of the substance, make up all losses, and vaporize the moisture; (2) the amount of air required to carry off the vapors after the temperature of the air has been lowered by the heating of the material and evaporation of the moisture; (3) the air and gases must never be heated to such a temperature as will injure or ignite the material; (4) the discharge temperature of the air must be such that the vapors will not condense by coming in contact with the material; (5) high temperatures of the incoming and outgoing air will increase the capacity as they will raise the temperature difference between heating medium and material, and also the carrying power of the air, but they will decrease the efficiency; (6) the fuel or steam consumption of a dryer is lowest when the outgoing air or gas is discharged at a temperature of about 176°F., and a saturation of from 80 to 90 per cent, and under otherwise equal conditions the efficiency is practically the same: (*a*) In a dryer where hot gases enter at 1,200° and more, and are discharged at 170° and 80 per cent saturation; (*b*) in a dryer where the air is used over and over again and only a part is discharged at 170° and 80 per cent saturation; (*c*) in a dryer which consists of several individual chambers, and where the air is re-heated repeatedly but finally discharged at 170° and 80 per cent saturation; (7) wherever possible, the heating surface should be placed inside the dryer to avoid losses and the waste air and gases must be removed as quickly as possible. The location of the discharge opening is also important and should be determined by calculation or experience.

More complete information, tables and calculations of capacities and efficiencies of such dryers will be found in the books of Hausbrand and Marr. All the data and constants should be used with caution as even a slight change in the process of manufacture will influence the conditions under which the material may be treated in a dryer. For this reason the manufacturer of drying machinery will usually ask for a sample of the material to be dried, and will only submit proposals after laboratory experiments and tests.

VARIOUS TYPES OF HOT AIR AND GAS DRYERS

Drying chambers are generally used for small quantities, and in cases where most of the work is done over night, without supervision. Fuel and steam consumption is therefore rather high. Figure 21 shows a standard chamber dryer with upward direction of the air. Figure 22, the same type with downward movement, which is some times desirable to reduce the speed of the air



FIGS. 21, 22, 23.—Chamber dryers with upward air current.

FIG. 24.—Tunnel dryer.

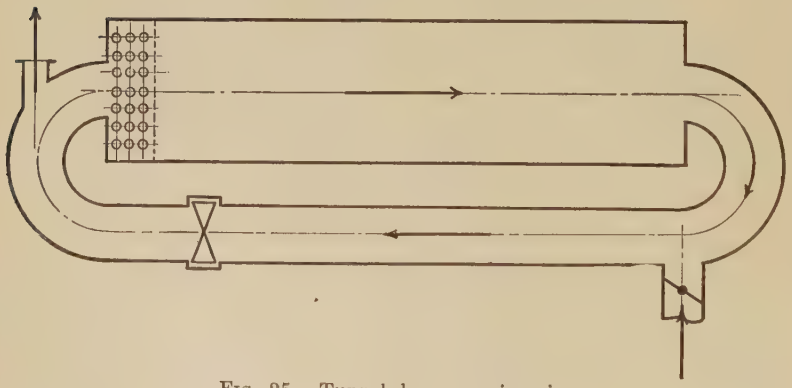


FIG. 25.—Tunnel dryer re-using air.

current, and avoid pockets and eddies. Figure 23 shows a dryer where the air is always reheated after each passage through one section; it is really a combination of a number of single dryers placed in series. Figure 24 describes a standard tunnel dryer, and Fig. 25 gives the same type with the provision that the air can be used over again. Figure 26 shows a special type where the air is cooled after every passage to remove the moisture by condensation. Such a dryer will require somewhat less steam, but the gain in efficiency is mostly lost by the extra expense for cooling water.

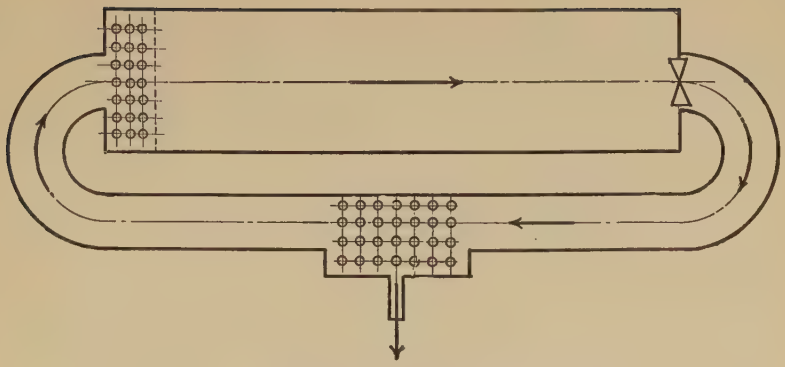


FIG. 26.—Tunnel dryer with moisture removal.

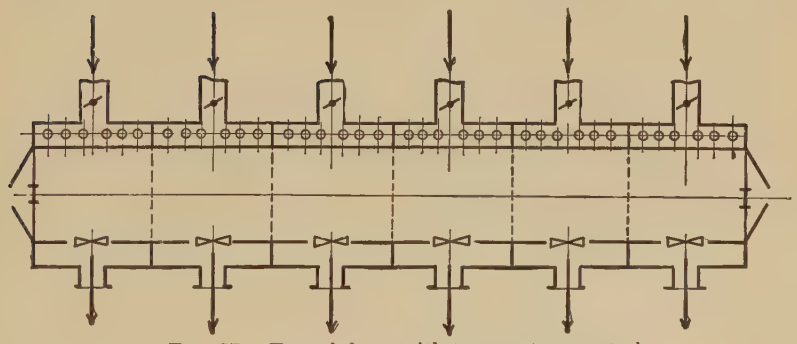


FIG. 27.—Tunnel dryer with temperature control.

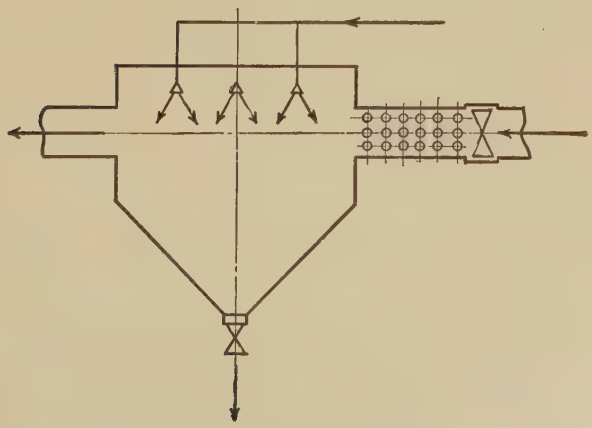


FIG. 28.—Spray evaporator.

Figure 27 shows an arrangement where the material is dried in different stages at various temperatures that can readily be controlled. The material can be handled in batches or continuously, and may be placed on trucks or slowly moving ribbons. This construction is mostly used for delicate substances that require certain temperatures for various stages of the drying process. It is in fact also a combination of a number of single dryers, combined to one unit in order to save time, radiation losses, and labor.

All these dryers will require from 1½ to 2 lb. of steam per pound of moisture evaporated, and are therefore not so economical as dryers heated by hot gases or indirect dryers heated by steam and hot water.

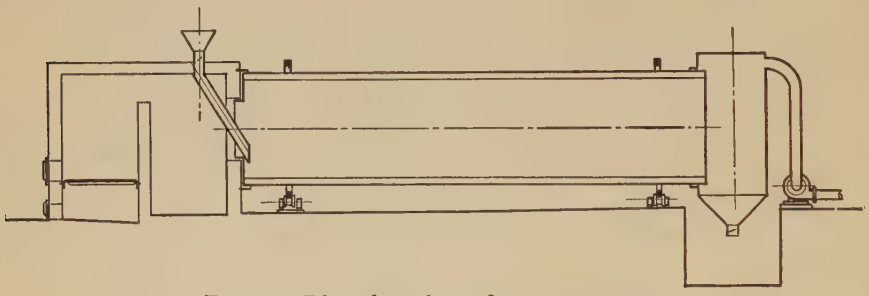


Fig. 29.—Direct-heat dryer, fire or waste gases.

So far, we have been discussing the treatment of solids, but there are also cases where it is desirable and necessary to convert a liquid as concentrated solution into solids in one operation. For this purpose it is essential that the liquid is divided into very small particles (atomized) before it is brought into contact with the hot air or gases. Such a fine spray is produced by discharging the liquid from spray nozzles under very high pressure, or by running the liquid on a horizontal disc revolving

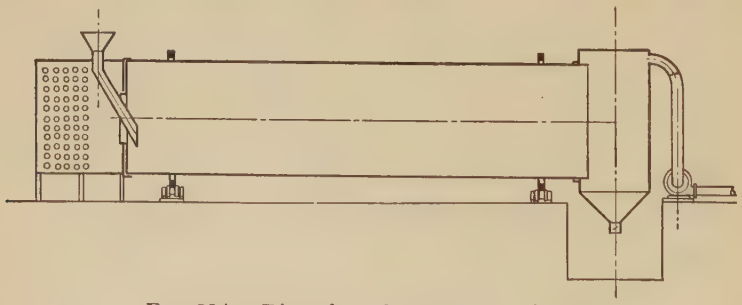


Fig. 29A.—Direct-heat dryer, steam or hot air.

at very high speed. Such a dryer is shown diagrammatically in Fig. 28. The air may also be cooled and reheated in the same manner as shown in Fig. 26. This type of apparatus is widely used for the drying of milk, but the steam consumption is high as compared with the vacuum-drum dryer, and losses of milk powder are unavoidable.

A direct-heat dryer using either fire gases or waste gases is shown in Fig. 29; the revolving drum may be plain as shown or have a second concentric tube for the return of the gases. Temperatures are generally high, and the material must always travel with the current of the gases and not counter-current to prevent scorching and igniting. The efficiency is usually much better than in the air dryers, and 1 lb. of coal will evaporate from 7 to 9 lb. of water. In some cases, cold air is mixed with the gases

to lower the temperature and the mixing should always be done after the combustion is completed to avoid fuel losses. Similar rotating dryers are also used for hot air, and frequently the interior of the drum is fitted with plates and baffles to afford a better contact between the air or gas, and the material to be dried.

A special type of dryer is shown in Fig. 30. It can be used for liquids or thin

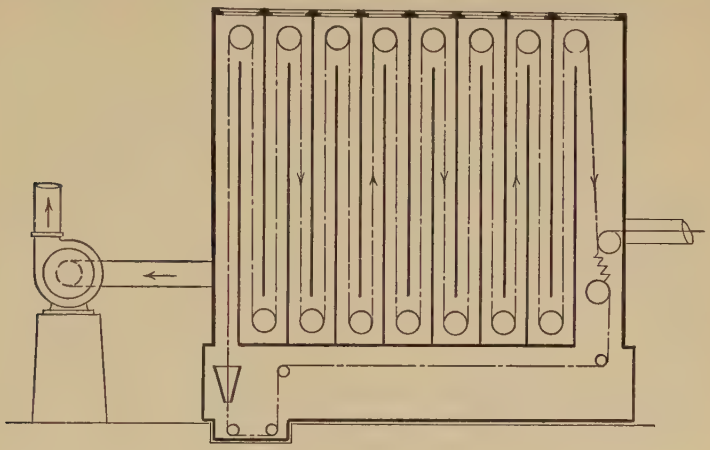


Fig. 30.—Dryer for liquids or thin pastes.

pastes only, and these are picked up by an endless wire apron which passes through the various chambers of the dryer continuously. After the drying process is completed, the material is removed from the flexible wire netting by breaker rolls and brushes. Generally the efficiency is good, as the fire or waste gases are utilized to the fullest extent on account of the long contact with the material.

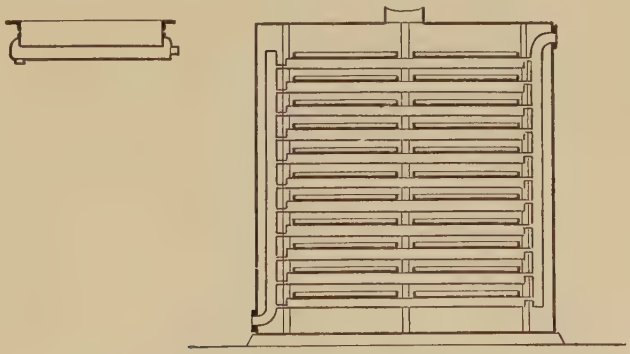


Fig. 31.—Pan dryer, steam-heated.

Drying by Steam and Hot Water.—In dryers of this type, the heat is transferred through metallic surfaces, and where liquids are to be dried, calculations may be based on the equations given in the previous chapter on "Heat Transmission." But where moisture is to be removed from solids, we have to resort again to experiments and tests in order to determine the capacity of a certain

size and type of apparatus. The drying may be done at atmospheric pressure, and in this case it is essential to remove the vapors by suction, as otherwise the condensation of these vapors will interfere with the drying process. The application of vacuum has many advantages, as it will lower the boiling temperature of the solvent, and reduce the steam consumption. However, vacuum dryers are expensive to install and operate, and can only be used for valuable products where a low temperature is essential.

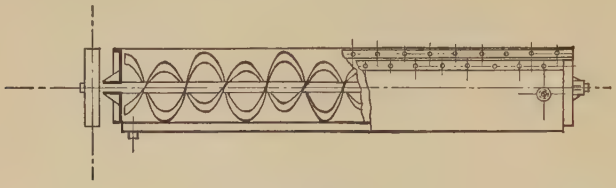


FIG. 32.—Trough dryer.

Various Types of Steam Dryers.—A simple apparatus is a jacketed shallow pan which is shown in Fig. 31. A number of such pans are frequently placed in one chamber, the outlet of which is connected to an exhauster or chimney. The material rests directly on the pan surface or is placed in metallic trays which can easily be handled. Figure 32 describes a trough dryer which may be used in batches or continuously. Here the material is agitated by paddles or screws and the capacity per square foot is therefore much higher than in tray dryers. Steam consumption is high, *i.e.*, from 1.6 to 2.0 lb. per pound of evaporation.

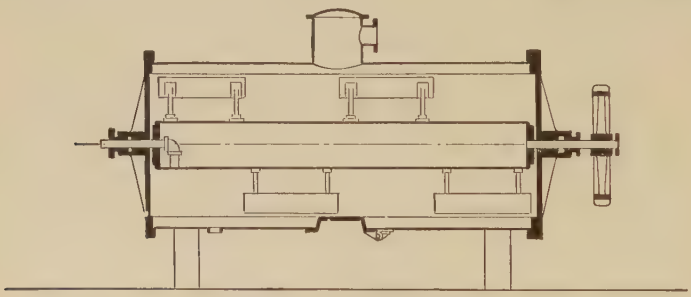


FIG. 33.—Rotary dryer.

Similar to the trough dryer is the so-called rotary dryer specified in Fig. 33. This may be used for either liquids or solids, and the capacity and efficiency are good. Both the outside jacket and the interior drum are heated by either steam, hot water or oil, and the heating surface is kept clean by the scrapers attached to the interior revolving drum. From 1.3 to 1.5 lb. of steam are required to evaporate 1 lb. of water.

Figure 34 shows a tubular dryer which can be arranged so that the steam or heating medium is inside the tubes, and the material on the outside, or *vice versa*. In either case it can only handle material of granular nature which will not stick to the surface. This type of apparatus gives generally a large capacity, as the heating surface can be arranged in compact form. The steam consumption is low, that is about 1.3 lb. per pound of water evaporated. A continuous plate dryer is specified in Fig. 35.

The hollow circular shelves are heated with steam, and the material enters at the top, and is discharged at the bottom. During the passage through the dryer, it is moved over each shelf by scrapers which are attached to a central shaft.

A machine very widely used in the chemical industry is the atmospheric drum dryer described in Figs. 36 and 36–A. Paper, all kinds of liquids and pastes are dried

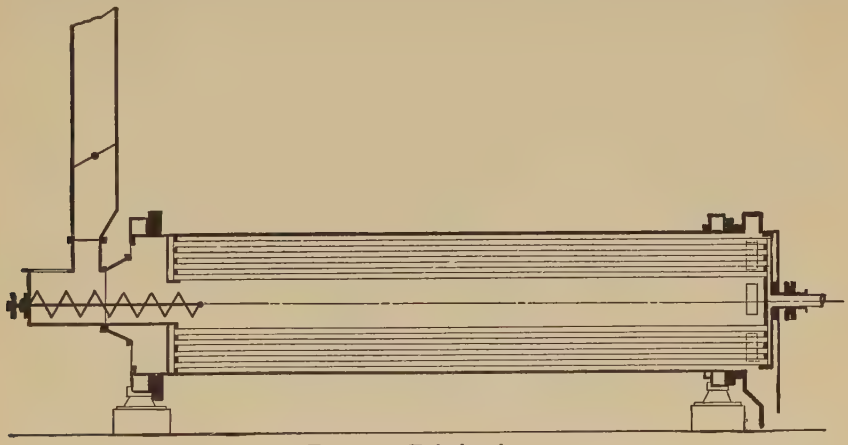


FIG. 34.—Tubular dryer.

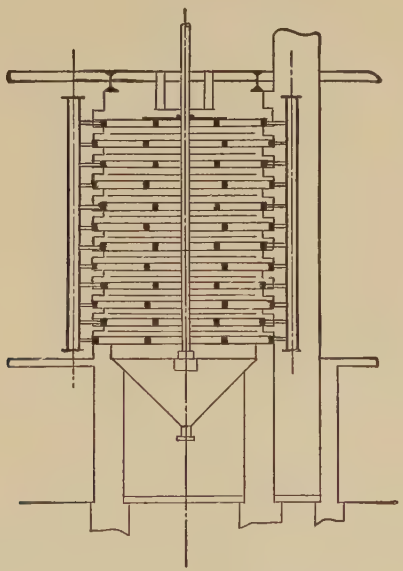


FIG. 35.—Continuous plate dryer.

on this machine, which can be arranged either in single or double rolls. The paste is fed to the drum from a tightly fitting trough, and for liquids, the drum either dips into a trough which is filled with this liquid, or the solution is sprayed against the lower part of the drum by pressure. The latter method gives a better distribution of the material, and therefore higher capacity of the dryer. In all cases, the capacity

is considerably higher than trough or rotary dryers, as the material is dried on the drum in a very thin film.

Drying Under Vacuum.—The most common type is the standard shelf dryer arranged very similarly to the apparatus shown in Fig. 31. However, the housings have to be built substantially in order to withstand the outside pressure. The vapors are condensed in either surface or barometric condensers. This type

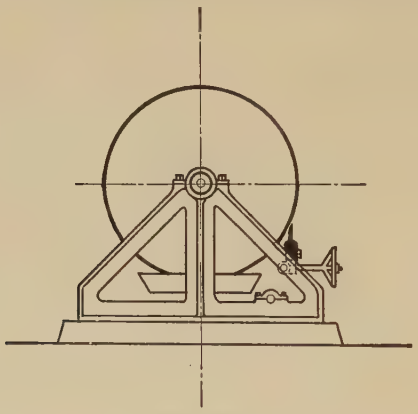


FIG. 36,

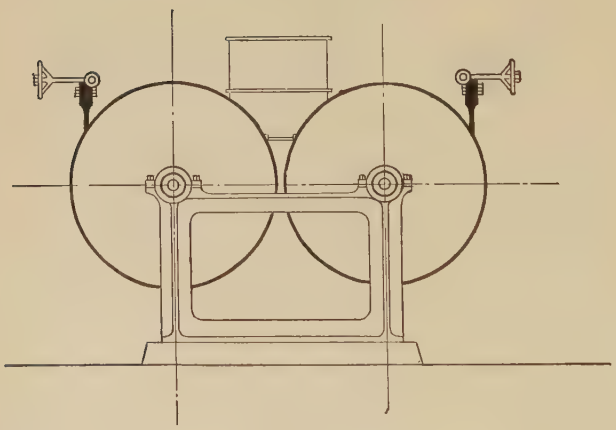


FIG. 36A.

FIGS. 36, 36A.—Atmospheric drum dryers.

is expensive in operation, as it requires much labor for the charging and discharging of the material in the trays placed on the shelves of the dryers. The rotary dryer shown in Fig. 33 is also used frequently under vacuum, and has given excellent results. A special type of apparatus is the ribbon dryer specified in Fig. 37, which really is a continuous shelf dryer. The liquid is spread evenly over the surface of the ribbon by special feed rollers, or spraying under high pressure. The ribbon itself consists of a fine mesh wire screen treated with a gelatinous coating, so that the liquid will never come in contact with the

heating shelves, upon which the ribbon is gliding. Such a dryer is extremely useful for the handling of sticky materials that have to be dried at low temperature. The work is continuous and the operation requires little attention and labor.

A vacuum dryer is shown in Fig. 18, and the construction is substantially the same as explained under the atmospheric drum dryer, except that the housing must be very substantial and air-tight. This type of dryer while rather expensive in first cost is economical in operation, and has given excellent results for the drying of milk and delicate extracts. In all vacuum dryers, the steam consumption is about 1.2 lb. per pound of water evaporated, and they have therefore a very much higher efficiency than the standard steam or hot air dryers.

Whatever type of dryer is used, it should be borne in mind that the material should be distributed over the heating surface evenly in small pieces and thin layers. Bulky material and large lumps will greatly retard the drying process, and while for instance a 2-in. board can be dried properly in from 2 to 3 days, it takes from 8 to 10 days to dry an 8-in. timber. The heat should be applied uniformly from all sides,

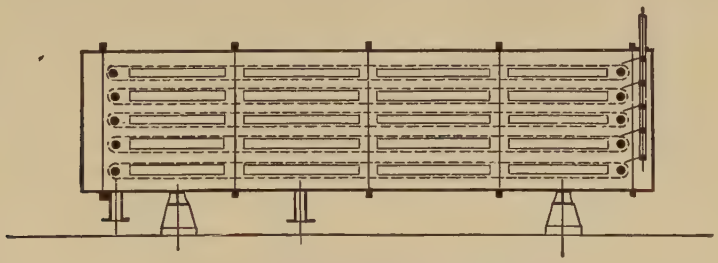


FIG. 37.—Ribbon dryer (vacuum).

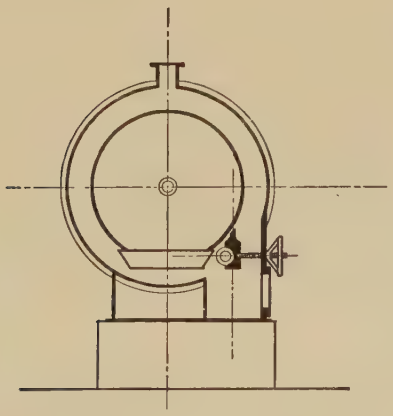


FIG. 38.—Vacuum drum dryer.

and the saturated vapors should be removed as quickly as possible in order that conditions may be as far as possible from equilibrium.

The drying process does not proceed at a uniform rate. Usually the first 80 per cent of the moisture can be removed in one-fourth to one-fifth of the total drying time,

but the hard work begins when the last traces of moisture are to be removed, that is when the material is wanted in a bone dry state.

PRACTICE OF DRYING

Organic Products.—Granulated sugar and table salt are usually dried in the rotary cylinder dryers heated by hot air. Sugar blocks are generally treated under vacuum. Organic extracts, milk, yeast, glues and gelatine, are handled in a vacuum-drum dryer, vacuum-ribbon dryer, and by the spray system. Glue and gelatin are also handled in chamber and tunnel dryers at low temperatures, which makes it necessary to cool the air artificially during the hot summer months.

Grain contains about 22 per cent moisture, and for storage this moisture must be reduced to about 10 per cent. The work is done on large drying floors and in towers, which are heated by hot air. A considerable saving of time and fuel can be accomplished by the use of rotary vacuum dryers. Textiles are generally handled in chamber and tunnel dryers, either in batches or continuous.

Rubber and electrical insulating material should be treated in standard chamber dryers or vacuum-shelf dryers, with or without the recovery of the so lvents. Shel dryers are more expensive to install, but the saving in steam quickly pays for the extra expense. Explos ves are handled in vacuum-shelf dryers, especially built for this purpose, with and without recovery of the solvents. The drying housing is equipped with large automatic doors which will open in case of explosion.

Paper and pulp are usually dried on atmospheric drum dryers heated by steam, and also in drying tunnels which are divided into several compartments that are kept at various temperatures.

Potatoes are first cooked, and the pulp is spread by rollers over the surface of large steam heated drying drums. Bituminous coal and all kinds of granular material are handled in tubular dryers. Soap is dried when in pieces in chamber dryers, and soap flakes are handled on ribbon dryers at low temperature. Starch, casein and similar products are handled in tunnel dryers and also in rotary vacuum apparatus. Food products, fruits and vegetables are usually dried by hot air in chamber dryers. Vegetables have also been handled under vacuum, which has given an excellent product.

Inorganic Products.—General rules for the application of certain types of dryers cannot be given as there are too many variations in the nature and quantity of the products to be handled. Usually rotary vacuum and drum dryers are used for delicate products, and direct-heated or steam-heated cylindrical rotary dryers are used for bulky materials that are not affected by higher temperatures. Chamber and shelf dryers (either atmospheric or vacuum) should be used for all delicate and sticky materials, also pastes, and in cases where the material is strongly acid or alkaline, as the trays that are placed on top of the shelves can be made of almost any kind of material which will prevent contamination of the product. Sticky material and paste can also be handled successfully on either a single or double roll drum dryer.

Waste Products.—Offal and tankage are generally handled in rotary steam jacketed dryers, in small quantities. Large plants use the tubular type, and the direct-heated dryer, and the concentrated stock coming from the tank-water evaporators is either added to the wet tankage, or dried separately on atmospheric drum dryers, so-called "stock rollers." Garbage which has first been degreased is usually handled in a direct heated dryer.

Beet pulp and brewers and distillers grain are treated in tubular rotary dryers with steam on the inside of the tubes. Beet pulp is also handled in direct heated cylinder dryers.

Hydrometer Equivalents

Liquids lighter than water

Degree, Baumé	Specific gravity	Degree, Brix	Degree, Baumé	Specific gravity	Degree, Brix
10	1.0000	0.00	56	0.7527	131.43
11	0.9929	2.86	57	0.7487	134.39
12	0.9859	5.71	58	0.7447	137.14
13	0.9790	8.57	59	0.7407	140.00
14	0.9722	11.43	60	0.7368	142.86
15	0.9655	14.29	61	0.7330	145.71
16	0.9589	17.14	62	0.7292	148.57
17	0.9524	20.00	63	0.7254	151.43
18	0.9459	22.86	64	0.7216	154.29
19	0.9396	25.71	65	0.7179	157.14
20	0.9333	28.57	66	0.7143	160.00
21	0.9272	31.43	67	0.7107	162.86
22	0.9211	34.39	68	0.7071	165.71
23	0.9150	37.14	69	0.7035	168.57
24	0.9091	40.00	70	0.7000	171.43
25	0.9032	42.86	71	0.6965	174.29
26	0.8974	45.71	72	0.6931	177.14
27	0.8917	48.57	73	0.6897	180.00
28	0.8861	51.43	74	0.6863	182.86
29	0.8805	54.29	75	0.6829	185.71
30	0.8750	57.14	76	0.6796	188.57
31	0.8696	60.00	77	0.6763	191.43
32	0.8642	62.86	78	0.6731	194.29
33	0.8589	65.71	79	0.6699	197.14
34	0.8537	68.57	80	0.6667	200.00
35	0.8485	71.43	81	0.6635	202.86
36	0.8434	74.29	82	0.6604	205.71
37	0.8383	77.14	83	0.6573	208.57
38	0.8333	80.00	84	0.6542	211.43
39	0.8248	82.86	85	0.6512	214.29
40	0.8235	85.71	86	0.6482	217.14
41	0.8187	88.57	87	0.6452	220.00
42	0.8140	91.43	88	0.6422	222.86
43	0.8092	94.29	89	0.6393	225.71
44	0.8046	97.14	90	0.6364	228.57
45	0.8000	100.00	91	0.6335	231.43
46	0.7955	102.86	92	0.6306	234.39
47	0.7910	105.71	93	0.6278	237.15
48	0.7865	108.57	94	0.6250	240.00
49	0.7821	111.43	95	0.6222	242.86
50	0.7778	114.29	96	0.6195	245.71
51	0.7735	117.14	97	0.6167	248.57
52	0.7692	120.00	98	0.6140	251.43
53	0.7650	122.86	99	0.6114	254.29
54	0.7609	125.71	100	0.6087	257.14
55	0.7568	128.57			

$$\text{sp. gr.} = \frac{140}{\text{Bé}^\circ + 130}; \qquad \text{Bé}^\circ = \frac{140}{\text{sp. gr.}} - 130$$

$$\text{Brix}^\circ = \frac{20\ \text{Bé}^\circ - 200}{7}; \qquad \text{sp. gr.} = \frac{400}{400 + \text{Brix}^\circ}.$$

The above formulas are based on Tagliabues' work, although the original Brix hydrometer was a saccharometer, arbitrarily graduated (see p. 398). Degrees Fisher are the same as degrees Brix.

$$\text{Beck}^\circ = \frac{17\ \text{Bé}^\circ - 170}{14}; \qquad \text{sp. gr.} = \frac{170}{170 + \text{Beck}^\circ}$$

$$\text{Cartier}^\circ = \frac{68\ \text{Bé}^\circ + 13}{70}; \qquad \text{sp. gr.} = \frac{136}{126.1 + \text{Cart}^\circ}$$

$$\text{Gay-Lussac}^\circ = \frac{5\ \text{Bé}^\circ - 50}{7}; \qquad \text{sp. gr.} = \frac{100}{100 + \text{G. L.}^\circ}.$$

Tralles' hydrometer is graduated from 0° in water at 60°F. to 100° in pure anhydrous alcohol (sp. gr. 0.79389) at 60°F., the graduation being such as to give percentage of alcohol by volume. Gay-Lussac's original hydrometer was apparently also an alcoholometer. Southworth's hydrometer (at one time a legal standard in New York) is an alcoholometer with its 0 at proof, and reading percentages above and below proof.

$$\text{Ballings' hydrometer; sp. gr.} = \frac{200}{200 + n}$$

Baumé's original hydrometer for liquids lighter than water was based on a zero point given by 10 parts of salt in 90 parts of water and 10° for pure water, the hydrometer stem then being divided in equal divisions. The 15° point on the "heavier than water" instrument is supposed to be the specific gravity of a solution of 15 parts salt in 85 parts of water Cartiers' hydrometer was probably only an infringement of Baumé's.

Hydrometer Equivalents

Liquids heavier than water

Degree, Baumé	Specific gravity	Degree, Twaddell	Degree, Brix	Degree, Baumé	Specific gravity	Degree, Twaddell	Degree, Brix
0	1.0000	0.00	0.00	37	1.3426	68.52	69.27
1	1.0069	1.38	1.24	38	1.3551	71.02	71.27
2	1.0140	2.80	4.04	39	1.3679	73.58	73.28
3	1.0211	4.22	5.82	40	1.3810	76.20	75.31
4	1.0284	5.68	7.63	41	1.3942	78.84	77.35
5	1.0357	7.14	9.40	42	1.4078	81.56	79.41
6	1.0432	8.64	11.22	43	1.4216	84.32	81.48
7	1.0507	10.14	13.01	44	1.4356	87.12	83.56
8	1.0584	11.68	14.82	45	1.4500	90.00	85.67
9	1.0662	13.24	16.64	46	1.4646	92.92	87.78
10	1.0741	14.82	18.44	47	1.4796	95.92	89.94
11	1.0821	16.42	20.27	48	1.4948	98.96	92.10
12	1.0902	18.04	22.08	49	1.5104	102.08	94.29
13	1.0985	19.70	23.90	50	1.5263	105.26	96.50
14	1.1069	21.38	25.74	51	1.5426	108.52	98.74
15	1.1154	23.08	27.56	52	1.5591	111.82	
16	1.1240	24.80	29.39	53	1.5761	115.22	
17	1.1328	26.56	31.21	54	1.5934	118.68	
18	1.1417	28.35	33.05	55	1.6111	122.22	
19	1.1508	30.16	34.90	56	1.6292	125.84	
20	1.1600	32.00	36.76	57	1.6477	129.54	
21	1.1694	33.88	38.62	58	1.6667	133.34	
22	1.1789	35.78	40.48	59	1.6860	137.20	
23	1.1885	37.70	42.43	60	1.7059	141.18	
24	1.1983	39.66	44.20	61	1.7262	145.24	
25	1.2083	41.66	46.08	62	1.7470	149.40	
26	1.2185	43.70	47.98	63	1.7683	153.66	
27	1.2288	45.76	49.88	64	1.7901	158.02	
28	1.2393	47.86	51.76	65	1.8125	162.50	
29	1.2500	50.00	53.68	66	1.8354	167.08	
30	1.2609	52.18	55.59	67	1.8590	171.80	
31	1.2719	54.38	57.50	68	1.8831	176.62	
32	1.2832	56.64	59.44	69	1.9079	181.58	
33	1.2946	58.92	61.38	70	1.9333	186.66	
34	1.3063	61.26	63.35	71	1.9595	191.90	
35	1.3182	63.64	65.31	72	1.9863	197.26	
36	1.3303	66.06	67.28	73	2.0139	202.78	

The above table is based on hydrometer readings made at 60°F., compared with water at 60°F

$$Bé° = 145 - \frac{145}{\text{sp. gr.}} \qquad \text{Sp. gr.} = \frac{145}{145 - Bé°}$$

$$Tw,° = 200(\text{sp. gr.} - 1) \qquad \text{Sp. gr.} = 1 + \frac{Tw°}{200}$$

The marine hydrometer for sea water has a range of 40°, the degrees expressing the third decimal of specific gravity, *i.e.*, 5° = 1.005 sp. gr.; 22° = 1.022 sp. gr.

The Brix hydrometer for sugar solutions is so graduated that 1° indicates 1 per cent of sugar in solution. The above table gives a comparison between it and the Baumé scale. In addition, if the specific gravity of a pure cane-sugar solution is known the above table will give the percentage of sugar in it.

Balling's hydrometer, much used by English dyers is so graduated that sp. gr. $= \frac{200}{200 \pm n}$, the + sign being used if the liquid is lighter than water, the − sign when heavier.

$$\text{Balling}° = \left(\frac{40}{29}\right) Bé°; \quad Bé° = \frac{29}{40}\,\text{Balling}°.$$

SECTION XI

CRYSTALLIZATION

By Arthur Given[1]

Crystallization is one of the oldest and also one of the most important processes applied to chemical products. It is one of the oldest, indeed perhaps the second oldest, since it can in many cases be brought about by the simplest of means, concentration due to natural evaporation. Concentration is probably the oldest chemical process known to man since it occurs naturally in the cases of natural solutions, *e.g.*, sea water in shallow rock pools, or in drying the juices of ripened fruits. Crystallization must be regarded as the second oldest chemical process as being the direct result of concentration. It may be contended that solution should be given first place for age, but it seems clear that this could only have been a man-controlled process *after* there were crystals prepared from natural solutions.

The term "crystallization" as used in this article is intended to cover not only the production of chemical compounds in the form of actual measurable crystals, but also the production from solution of crystallizable compounds desired, in any solid form, either crystalline or amorphous.

The great importance of crystallization lies in the three chief things which it accomplishes:

1. It produces the substance in a solid form, thus making it much simpler and easier to transport it and to preserve it.

2. It gives a great decrease in weight and usually in bulk, which further increases the advantages under (1).

3. It produces a material of a greater, usually much greater, degree of purity than the solution from which derived.

A large majority of chemical products as such, whether natural or synthetic, are first produced as solutions of greater or less purity and concentration. In many cases where these solutions are of sufficient purity, *i.e.* contain besides the solvent either no impurities which will interfere with later use, or only small amounts of such impurities, crystallization offers no advantages. A few cases may be cited:

(*a*) When the substance cannot be produced in the desired form or composition in the solid phase.

(*b*) When the substance is to be used as a solution in close proximity to the point of production.

(*c*) When it is not readily usable in the solid form.

It is, however, far more usual for chemical compounds which are solids in a pure state at normal temperatures to be finally manufactured, handled and sold in some form of crystalline condition.

Since crystallization invariably means more or less purification, even though by no more than the removal of the solvent, the ideas and processes of crystallization and purification are inextricably interwoven and it is impossible to consider crystallization

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without at the same time considering purification, especially since it is frequently impossible to secure crystals of a reasonable degree of purity, or even in some cases to get crystals of the desired product at all without resorting to some process of purification.

The word purification as used in future throughout this article should be understood to be limited to the removal at any stage of manufacture of substances other than that desired as the final product, excluding, however, the immediate solvent; and irrespective of whether the impurities are removed from the solution or whether the desired substance, or the material from which it is to be derived, is separated by some process, leaving the impurities behind. This being understood, it will readily be seen why, in considering means of crystallization, means of purification play such an important part as to really be means of crystallization almost in themselves.

The most important means of crystallization or basic processes of crystallization may be briefly stated as follows:

1. Concentration.
2. Concentration plus cooling.
3. Precipitation by some reagent as a comparatively pure salt of the substance desired, which salt can be readily separated and after separation can be readily decomposed, setting free the desired product in such a condition that the precipitating agent can be quite completely removed and the solution of the pure substance crystallized by concentration or other equally simple method.
4. Cooling.
5. Addition of a solvent for impurities in which the desired product is not, or is not readily, soluble.
6. Addition of some substance which precipitates the desired product in a more or less crystalline form, either as applied to original solution, to a concentrated solution, or by concentration after addition of precipitant.

These means of crystallization will be discussed at length with various modifications and amplifications.

It is necessary for the sake of clearness in describing a process to consider its application to some well-known case. Because of the fact that sucrose (cane sugar) is one of the oldest chemical compounds to be manufactured and at the same time one which requires many operations to obtain in a pure crystalline form, there has perhaps been more chemical ingenuity expended upon its manufacture than upon any other product; and there are today in actual use more processes of purification and of crystallization applied to this article of commerce than to any other. There are practically no processes of purification and crystallization in use which are not typified in some of the processes of sugar manufacture. For this reason the production of this substance will be cited in a very large percentage of the processes outlined, with discussion of similar methods as applied to entirely different cases. This is not to say that these processes originated in the sugar business, for most of them were devised for other uses and adapted to sugar making. It is only that long experience has brought about a fair degree of knowledge of the multitude of methods used, and that more cases can be covered in this direction than in any other.

Concentration.—The oldest, simplest and, combined with cooling, most used method of producing crystalline solids from solutions is by concentration of the solution. The oldest application is without doubt found in the evaporation of sea water for the production of sodium chloride (common salt). Prehistoric man must have learned this process from the accumulation of salts left by the evaporation of shallow pools of sea water above tide level filled during storms and concentrated to the point of crystallization by combined sun and wind. Except in so far as the best method for securing crystals of highest purity has been

developed, this process is today carried out in many parts of the world in exactly the same manner as it has been for thousands of years. Shallow tanks or reservoirs, often of enormous size, are pumped full of sea water, which is allowed to concentrate naturally until the crystals of salt begin to form and collect in considerable quantities, when they are skimmed out and a new crop allowed to form, to be removed in turn, the process being allowed to continue until some of the other salts begin to separate, when the mother liquor is run off and the tank refilled.

Since the evaporation is mainly due to the heat of the sun the process is largely confined to the warm countries where the percentage of bright days is high. This is practically the only process of crystallization which is carried out without artificial assistance and is of special interest not only for that reason but also from the fact that it is a most beautiful example of purification by crystallization, there being a large number of salts other than sodium chloride in sea water. In this process of natural crystallization the salt crystals of high purity are produced due to the fact that the combination of other salts is somewhat more soluble than the sodium chloride, and that there is a greater amount of it, so that it accordingly crystallizes out first as the concentration proceeds.

The production of sodium chloride by the evaporation of natural brines is of great antiquity in China. These brines contain very small amounts of impurities as compared to sea water and in Chinese practice are generally evaporated to solid cakes in iron pans. In most other countries, however, the demand is for a salt of high purity. The brines are therefore concentrated in some form of evaporator and the crystallization allowed to take place progressively, the crystals being worked in one direction and mother liquors in the other. Several grades of salt are made for various uses according to the degree of purity desired. The purest crystals go for table and dairy use, others for cattle and other stock, for manufacturing, etc., down to the most impure grades which are made from the final mother liquors largely for use in freezing mixtures for ice cream, etc.

In this country within the last few years an important improvement in process increases the yield of high purity crystals at the expense of those of low purity. Many salt wells contain measurable quantities of barium salts, which are decidedly poisonous and have caused much loss through the use for stock of low purity salt containing salts of barium. The improvement in process is purification by precipitation of impurities. The brine as it comes from the wells receives a calculated quantity of salt cake (acid sodium sulphate), which, being a byproduct of nitric acid manufacture, is very cheap. The sulphuric-acid radical combines with the barium salts to form the insoluble barium sulphate, which is removed by a settling process and the practically barium-free brine concentrated as usual. Crystallization by concentration has been treated thus fully because it is a somewhat rare case.

Sodium chloride is peculiar in that it is little more soluble in hot water than in cold. Salts in general show greater solubility as the temperature of the solution rises though there are cases, notably that of some calcium salts, in which the solubility decreases as the temperature rises. The first case is so generally true that crystallization by concentration and by cooling may be regarded as steps of the same process. However, crystallization by cooling alone has its own distinct place as will be described later.

Crystallization by Concentration and Cooling.—There are very many cases in chemical manufacture where the desired product is produced as a practically pure water solution of the substance. This is especially true of metallic salts such as ferrous sulphate, copper sulphate, zinc sulphate, lead acetate, etc. The solutions of these salts are generally concentrated in open evaporators, usually

wooden tanks, to the proper density for crystallization, then run into crystallizing tanks and allowed to cool and deposit crystals until no more material will crystallize out. In one large plant the acid ferrous-sulphate solution which has been used for removing scale from wire rods is boiled with iron scrap until the acid is completely used up, then run through filters to remove mechanical impurities, and allowed to cool and crystallize in wooden tubs 12 ft. in diameter by 3 ft. deep. When crystallization is complete the mother liquor is drained from the bottom of the tub and returned to the process. The crystals, frequently 18 in. deep, are shoveled out into cars where they drain and dry. They are then ready for use or shipment.

In another large plant making copper sulphate and zinc sulphate, the solutions are concentrated in open lead-lined evaporators. The crystallizing tanks in this plant are very large, 5 ft. wide by 2 ft. deep by 60 ft. long. In order therefore that the crystallization may be uniform the batches of concentrated liquors are run into large storage tanks where the liquor is kept hot until enough is accumulated to fill one of the crystallizing tanks. Upon completion of crystallization the mother liquor is drained off and the crystals shoveled out on to an inclined draining platform, which permits the drainage from the crystals to run back into the crystallizing tank. The crystals remain upon the draining platform until dry when they are shovelled into cars and taken to the packing house.[1]

Crystallization by Agitation of Concentrated Solutions During Cooling.—The above plant produces two grades of zinc sulphate, one the large clear crystals for ordinary commercial use, the other a fine granular crystal such as is found in drug stores and used where superior purity is desired. This type of crystal seems likely to supersede the large irregular crystals of all kinds with which we are most familiar. It has already done so almost entirely in the case of sodium bichromate and has been found satisfactory in the case of sodium ferrocyanide (yellow prussiate of soda). These small crystals are produced by artificially cooling the concentrated liquor, at the same time keeping it in constant motion. Under these conditions only the small crystals form. The process is much more rapid than that usually followed, since by the time the liquor is thoroughly cold nearly all the possible crystals have been thrown out, and being so small are much purer than the large crystals. The saving in space, size of equipment and time of production offset the cost of cooling the liquor; and in the best equipped plants using this process it is very nearly continuous, the cold liquor being so stirred that the crystals are kept in suspension and go direct from the crystallizing tank to the filter or centrifugal machine, whence the dried crystals go to the packing room and the mother liquor and wash are returned to process for reconcentration and recooling.

The material of which tanks are made has some influence on this point of quick crystallization. The ordinary tanks for copper-sulphate crystallization are of wood, lined with lead. It has been found that the solution will cool much more quickly in metal tanks lined with lead, and that even cutting "windows" in the wood work of the tank, where it is structurally safe to do so, helps quick crystallization. This, though, was not an advantage in former times, as the trade paid fancy prices for large

[1] In making double salts, it is probable that better results are obtained if both components are put into solution separately rather than that one should be dissolved in a concentrated solution of the other. The observations may be baseless, but I have always thought for instance, that superior crystals of nickel-ammonium sulphate were produced by bringing both single sulphates into solution separately. —EDITOR.

copper-sulphate crystals, so that strips of lead were hung in the tanks on which the largest crystals deposited (see remarks concerning "rock candy," p. 404), and the attempt was made to get as great a crop as possible on these strips.

The material that formed at the bottom and lower portions of the sides of the tank was known as "bottom crystal." The amount of this inferior material was increased by the "windows," although interest on capital was saved by the shortened time of crystallization.

On the other hand, the time of crystallization was increased, and the relative amount of bottom crystals diminished, by carefully floating about an inch of water on top of the sulphate mother liquor just after dropping the liquor into the tanks. The water had a less radiating power (or else a lower coefficient of conduction) than the mother liquor, and materially slowed down the drop of temperature in the early hours of the crystallizing process.

In this connection it may be noted that trade prejudices must be considered in crystallizing. Some ferrous-sulphate (copperas) crystals that were far purer than the trade had been used to were unsalable because the trade (at that time) was used to a crystal with some brownish stains and these were all a beautiful green! Happily time cured their too brilliant appearance.

Crystallization in Motion.—The highest development of the crystallization in motion process is in sugar factories. Because the juices of sugar-producing plants contain many substances other than sugar, only a part of which can be commercially removed, the production of sugar of the purity to which we are accustomed, 99.7 to 99.9 per cent, is necessarily a complicated matter.

Before the introduction of the "crystallization in motion" process cane-sugar crystallization was carried out in three stages. The juice, properly "clarified" by heating with milk of lime followed by settling and separation of the clear juice from the sludge, is concentrated in multiple-effect evaporators to "thick sirup" or "meladura," in which form it is taken into the vacuum pan for final concentration. By careful adjustments of heat and vacuum the formation of crystals is brought about. When the crystals have grown to the desired size the "massecuite" is discharged into iron "sugar wagons" in which it stands until cold, when it is dumped into mixers and when of uniform consistency is run by gravity to the centrifugal machines where the molasses is expelled. The crystals are "first sugar" of 94 to 96 per cent purity which goes upon the market as raw sugar for refining. The molasses is "first molasses" and is reconcentrated in vacuum pan, cooled and centrifuged as above. The crystals are "second sugar" and are melted in fresh juice and recovered as "first sugar." The molasses is "second molasses." It cannot be boiled to "grain" or so as to form crystals, but is boiled to a consistency called "string proof" and stored in sugar wagons for from 2 to 6 months, usually until just before the beginning of the next campaign when it is centrifuged and the "third sugar" worked up with fresh juice. The "third molasses" is the final cane molasses and is of 27 per cent to 33 per cent "purity," which is the per cent of sugar in the total solids present.

In the most modern process the "thick juice" is boiled to grain as above described, but the massecuite instead of being discharged into cars is run into huge "crystallizers" 8 ft. in diameter by 30 ft. long, lying horizontal, and provided with powerful stirring arms on a horizontal shaft. These crystallizers are usually provided either with a water jacket or with cooling coils for controlling the temperature. The separation of the sugar from the viscous mass (massecuite) of crystals and molasses is a very different problem from crystallization from relatively pure water solutions as in the cases

previously cited. The great mass in the crystallizer cools slowly, the sugar separating and building up on the already formed crystals. From 8 to 10 days are required to cool completely, and frequently the cooling has to be assisted by cooling water, especially in tropical countries. The cool massecuite is pumped by large open cylinder "magma pumps" to mixers which deliver it by gravity to the centrifugal machines.

The first molasses is boiled to grain as above, and the second massecuite is put through the crystallization process described. The long cooling with constant stirring carries the separation of the sugar as far in two operations as the older process in three, the second molasses being of 25 to 28 per cent "purity." Moreover, another improvement does away with the melting and recrystallization of the second and third sugars. The second sugar, from which all possible molasses has been expelled, is placed in a mixer with fresh first molasses and after thorough mixing is again centrifuged. The expelled wash molasses goes direct to second vacuum pan. The molasses free crystals are "washed up" with water to a test of 94 to 95 per cent and go on the market as standard raw sugar. The wash goes to the first pan for first sugar. Only part of the first molasses is needed for mixing with the second sugar, the excess going to second vacuum pan. The many advantages of the modern process are evident.

A curious analogy to the boiling to "string proof" of third molasses is found in the manufacture of chrome alum. It is well known that if molasses be boiled to a high temperature and allowed to cool "taffy" results, a vitreous noncrystalline mass. In like manner if in concentrating a solution of chrome alum, the evaporation passes a certain point, the mass on cooling assumes a glue-like consistency and refuses to crystallize. In such a case it is necessary to redissolve and reconcentrate carefully.

The manufacture of refined sugar from raw sugar is carried out in a manner very similar to the former practice for raw sugar, since quality is absolutely essential. It differs from that described in that several successive crops of crystals are taken from the liquor and that frequently, when the liquor is too impure to yield further crops of white sugar, several grades of "soft sugars" are made. When "soft sugars" are not made the impure liquor is grained at a higher temperature and the second and third sugars are remelted and returned to process for white sugar. The final molasses from raw-sugar manufacture is fit only for cattle feed or for alcohol. The final molasses from the refining process is known as "refiner's sirup" and is well known as the flavoring ingredient of the commercial so-called "corn sirups."

Deep-tank Crystallization.—The manufacture of "rock candy" from refined sugar is of interest as presenting a survival of ancient practice of crystallization formerly used for chromates and bichromates and still in use in some plants producing ferrocyanides. The concentrated liquor is run into rather deep tanks in which are suspended a large number of strings. As the liquor cools the crystals form on the strings and when crystallization is complete, usually in 3 to 5 days, the mother liquor is run off and the crystals stripped from the strings in the case of large commercial production, while for a strictly quality production such as "rock candy" threads are used, which are allowed to remain in the crystals.

This process if carried out with solutions of moderate concentration, cooled slowly, produces crystals of maximum size and beauty. The strings, however, are a constant source of trouble from tangling and from breaking during stripping. For this reason in some plants wooden strips $1\frac{1}{4}$ by $\frac{3}{8}$ in. by 4 ft. long have been substituted with good results. The crystals form as readily as on the string and are readily removed by pulling the sticks through a slot in a piece of flat iron bolted to the inside of a box for collecting the crystals.

Precipitation and Decomposition Preparatory to Crystallization.—The manufacture of sugar from sugar beets is a more complicated problem than its manu-

facture from sugar cane because of the much larger amount and difficulty of removal of the impurities. The treatment of the beet juice is much more difficult than that generally applied to cane juice, although in cane factories making white sugar the process is practically the same as in beet factories. The boiling and treatment of thick juice is practically the same for beet factories as for cane factories so far as the first crops of crystals are concerned. The interest of this section in beet-sugar manufacture lies in the methods employed in getting a maximum yield of sugar from the molasses. Owing to the quantity and nature of the impurities in beet juices the final beet molasses has a "purity" of approximately 50 per cent as against 25 to 30 per cent for final cane molasses. Moreover, the beet molasses is of such a character that it has little value, except by special treatment, for the uses to which cane molasses so readily lends itself. This being the case, coupled with the fact that on account of its high sugar content there is a much greater quantity of final beet molasses produced than is the case in cane factories, it has been found necessary, in order to produce beet sugar successfully, to devise means of recovering as far as possible all the sugar in the molasses.

Three processes have been worked out and extensively used. In order of their success as chemical processes they are: (1) the osmose process in which the crystallizable materials in the molasses are separated from the non-crystallizable by osmosis in cells with permeable diaphragms; (2) the Steffens or lime process, in which the sugar is precipitated as calcium trisaccharate; (3) the barium process, in which the sugar is precipitated as barium trisaccharate.

Purification by Osmosis.—The osmose process was the first commercially applied and was at one time extensively used in Europe though only a little in this country. It has been almost entirely superseded by the Steffens process. The development of the osmose process was a great step in advance, but it has two inherent objections which caused its disuse soon after the Steffens method was brought out. The first difficulty is that for successful osmosis relatively dilute solutions must be used, which increased largely the amount of evaporation necessary to recover the sugar. The second objection is that not only the sugar, but also all crystallizable salts in the molasses pass the diaphragms, so that while this process made it possible to recover a large amount of the sugar in the molasses, the concentration of salts prevented the crystallization of a great deal of the sugar, this being a peculiar property of many of the mineral salts commonly found in molasses. A third, though less important objection, common to both the osmose and Steffens processes, is that the sugar raffinose (melitose, $C_{18}H_{32}O_{16} + 5H_2O$), which is found in considerable amount in beets under some conditions, is separated along with the sucrose and remains to upset calculations and give trouble later. Nevertheless the process offers a means of separating crystalloids from colloids especially where a battery of dialyzers is used.

Calcium and Barium Sucrates.—The formation of insoluble compounds of sucrose with salts of the metals of the alkaline earths, calcium, barium and strontium has long been known. The use of salts of barium and strontium commercially was proposed and tried out in Europe many years ago but failed to receive approval, perhaps largely on account of the fear of traces of the known poisonous salts in the finished product. The cost of the necessary salts and the difficulty of reconversion into usable form also had great commercial influence. The value

of the general method was, however, well recognized and almost as soon as Steffens perfected his process of using lime, it sprang into nearly universal favor.

In carrying out the Steffens process, the diluted beet molasses is added with thorough mixing to a thick cream of freshly slaked lime (calcium hydroxide). Two important reactions take place. The sucrose forms the insoluble calcium trisaccharate, and the excess lime destroys any reducing sugars which may be present, and which, if separated with the sucrose adversely affect crystallization. The lime-sucrose precipitate is filter pressed and washed free of soluble salts, mixed to a stiff cream with water and the sugar set free by passing carbon dioxide into the mixture forming calcium carbonate. The sugar solution is filtered off, the calcium carbonate washed free from sugar and reburned to furnish lime and carbon dioxide for further use. The sugar solution is evaporated and crystallized along with fresh beet juice. The washings are used for mixing the lime-sucrose cream preparatory to carbonation. There are two objections or rather difficulties with the Steffens process. The first is that it is difficult to maintain the exact conditions under which all the sucrose will be precipitated, so that there is apt to be some loss of sugar in the washings of the lime-sucrose precipitate. The second difficulty, already mentioned in connection with the osmose process, is that raffinose is equally as precipitable as sucrose and is therefore carried back into process with the sucrose. The result is that every so often the raffinose content of the molasses reaches a limiting maximum and a batch must be thrown away with the unavoidable loss of the sucrose contained.

There are plants abroad using both the strontium and the barium processes, but so far as known only one in this part of the world. This is a Canadian plant using barium carbonate as raw material. Because of the difficulty of driving off the CO_2 it is necessary to heat the barium carbonate in an electric furnace. The barium oxide converts readily to barium hydroxide on treatment with water, and is used as a strong solution. The addition of the beet molasses to this barium-hydroxide solution instantly produces a thick precipitate of barium trisaccharate, and by working at the correct temperature the formation of the raffinose salt is prevented. The precipitation of the sucrose is readily made complete and a recovery of 95 to 97 per cent of the sucrose in the molasses is claimed. The decomposition of the barium-sucrose precipitate is so complete that not a trace of barium is normally found in the sucrose solution. A breakage of filter cloth may sometimes allow a little barium carbonate to escape, but this is prevented from getting into process by a system of settling tanks. In spite of the high cost of the conversion of the barium carbonate into barium hydroxide, the high percentage of recovery makes the process financially profitable even when compared with the Steffens process. An interesting side light is the fact that the filtrate from the barium-sucrose precipitate is concentrated and sold for the manufacture of potassium salts for fertilizer for growing new crops of beets.

Crystallization and Purification of Ferrocyanides.—Closely allied to the above described processes for sugar is the purification and crystallization process for ferrocyanides. After leaching out the soluble ammonium salts present the insoluble iron-cyanogen compounds in the spent oxide from purifying coal gas are converted into the soluble lime salt, calcium ferrocyanide, by treatment with slaked lime and leaching. The resultant solution contains a variety of undesirable compounds, chief of which are soluble sulphates and sulphocyanides. Calcium ferrocyanide has the property of forming a very slightly soluble salt with ammonia in the presence of hydrochloric acid. In practice, in the manufacture of sodium ferrocyanide, to the calcium-ferrocyanide solution is added the crude ammonium-sulphate solution previously leached out, hydrochloric acid is added to acidity and enough ferrous sulphate to complete the precipitation.

The mixed precipitate of double salt and ferrous ferricyanide is filtered out, washed to remove sulphates and sulphocyanides and decomposed by boiling with sodium carbonate, forming the sodium ferrocyanide and setting free the ammonia which is distilled off and collected. The sodium ferrocyanide solution is filtered off from the iron-lime precipitate, concentrated and crystallized as already described on page 404. Unfortunately the double salt precipitate is very difficult to wash, with the result that frequently considerable amounts of sulphates pass into the finished liquor as sodium sulphate. The first crop of crystals is normally a pure, high-grade sodium ferrocyanide. The first mother liquor is concentrated to proper density and crystallized in like manner.

If the washing of the double salt has been at all thorough the crystals from the first mother liquor are of good grade but if not they are added to first or fine liquor and recrystallized as first-grade product. The second mother liquor is in turn concentrated and crystallized, the crystals going back into the next batch of first mother liquor and the third mother liquor being again concentrated and crystallized. These crystals go into second mother liquor and the fourth mother liquor which usually contains not more than 3 to 5 per cent ferrocyanide is either discarded or worked up for its sulphocyanide content, by this time often as high as 25 to 30 per cent. All first-crop crystals are dried in centrifugal machines and washed with steam and are ready for packing and shipment. In plants where strings are used for crystallization, the strings are pulled out where possible, the large bodies of crystals rinsed free from mother liquor, drained and air dried. Crystals to be returned to process are simply stripped and drained.

Methods 1, 2, and 3 (p. 399) cover the most common and most important means of crystallization. Method 4 (simple cooling) by itself is a special case, more applicable commercially to organic compounds than to inorganic. A typical example of its application is in the separation of isomeric compounds having different melting points, *e.g.*, separation of meta dichlorbenzol from paradichlor benzol. The meta compound boils within 3° of the para and is completely soluble in the para compound, the meta being, however, a solid at ordinary temperatures while the para is a liquid. By cooling the solution of meta in para to 0°C. the meta nearly all separates in beautiful crystals and the para can be drained off nearly pure. This method has a fairly wide application in similar cases and will solve many troublesome problems of separation.

Method 5 (addition of a solvent for the impurities) is of technical interest though as yet of only limited commercial application. It has been used principally in the preparation of some of the rarer sugars, many of which are very difficult to crystallize from the solution in which they are prepared exactly as is the case in recovering sucrose from molasses. It has been found that by addition of glacial acetic acid the impurities are made more soluble while the sugars crystallize out readily. Cases are known where concentrated solutions have stood for months without developing crystals, which after mixing with glacial acetic acid, produced a crop of crystals of unusual purity over night. Like many other chemical processes, the principle of the operation has been known for many years, but its practical application has only recently been made, and even now has not been greatly extended.

Method 6 is another rather special case, which, however, finds extensive commercial application in certain lines. At first sight it may be thought to be no more than an application of precipitation but precipitation is generally brought about by a reaction between the precipitating agent and substance in solution to form a new compound. Rather Method 4 is intended to apply to cases such as are represented by separation of compounds by immiscible solvents, the compound, or the impurity as the case may be, being soluble in both solvents, while the body to be separated is soluble in only one. Method 6, however, applies to many cases where the separated body comes out in

solid form. Perhaps the most familiar example is the "salting out" of soap, where common salt is added to thick soap solution in the boiling kettle, causing the soap to separate from the water and lye and to collect in dense masses which can be worked up as desired. An even better example is the case of some of the nitro compounds which are soluble in the strong acids in which the nitration takes place but upon drowning the acid in water appear as small but well defined crystals. A very similar case is the precipitation of organic compounds from solution in fine crystalline form by addition of solvent which dissolves in all proportions in the solvent holding the desired material but in which the material itself is not soluble, *i.e.*, the precipitation of pure sugars from water solution by strong alcohol.

Another case is the precipitation of silver nitrate from strong nitric-acid solutions, and of barium chloride from strong hydrochloric-acid solutions.

Notwithstanding the vast production of crystalline products and the great amount of work which has been done on crystallization, our knowledge is still to a large degree empirical. If the laws of physical chemistry are applied to crystalline chemical products it is possible to get better and more uniform results than are now generally obtained, especially in crystallization from solutions of several components. The practical application of these laws will of course require close control and more expensive equipment and means for maintaining temperatures, but the results will be more than commensurate. An excellent example of what is possible in this direction has been the development of the Frecth or Brunner-Mond process of the manufacture of ammonium nitrate from ammonium sulphate and sodium nitrate. In this process the four-component system $(NH_4)_2SO_4$ — $NaNO_3$ — NH_4NO_3 — Na_2SO_4 was studied and worked out in the laboratory by the phase rule and the exact concentrations and temperatures necessary to bring about the transposal were determined and then put in practice. The plant erected in this country for supplying the need of ammoniumnitrate during the war had a capacity of 600,000 lb. per day of ammonium nitrate of 96 to 97 per cent purity. A general outline of the process is as follows: A hot solution of ammonium sulphate of exactly the predetermined density was made and to it was added a calculated quantity of sodium nitrate. With thorough mixing and cooling to the exact extent necessary the bulk of the sodium sulphate formed and separated out and was removed from the solution by continuous filters. The filtrate was diluted to the density for the proper crystallization of the ammonium nitrate and run into long shallow crystallizing pans arranged in tiers in an immense room the air of which was constantly circulated and kept at exact temperature and moisture content which had been determined as most favorable for ammonium-nitrate crystallization. When the proper time for crystallization had elapsed, the mother liquor was drawn from the pans and the crystals shovelled out, dried, and shipped.

The sodium sulphate crystals were washed with a little water and again filtered, the crystals being dumped out and the wash going to join the mother liquor. This mixed liquor was concentrated to the proper point to receive more ammonium sulphate and another charge of sodium nitrate, and the operation continued. The crystallizing rooms contained some miles of pans and had the largest air-conditioning plant ever erected. It has been stated that by the addition of a few units in one or two steps of the process the capacity could readily have been increased to 800,000 lb. per day.

A proper application of phase rule control would greatly simplify the problem of handling impure solutions such as described under ferrocyanides, raw brines etc., and doubtless many others which it has not been attempted to discuss. It seems to be a method of wide potential application but necessitates considerable laboratory work.[1]

[1] In connection with this chapter one should also study Sec. IX on leaching and dissolving; Sec. X on evaporation; Sec. XXVI, the portion covering the preparation by crystallization of radium compounds; the tables of solubilities in the appendix and the Baumé-Brix-Twaddell-specific gravity conversion tables on pp. 397 and 398.

SECTION XII

PYROMETRY

By Paul D. Foote, C. O. Fairchild,
T. R. Harrison and R. M. Wilhelm[1]

The High-temperature Scale.—The ideal temperature scale is known as the thermodynamic scale. Kelvin has defined this scale to be such that "the absolute values of two temperatures are to one another in the proportion of the heat taken in to the heat rejected in a reversible thermodynamic engine working with a source and refrigerator at the higher and lower of the temperatures respectively." The size of the unit degree is given by arbitrarily defining the temperature interval from the freezing point to the boiling point of pure water under standard conditions (760 mm. Hg. pressure) as equivalent to 100 Centigrade degrees.

No thermometer or pyrometer uses this principle of temperature measurement but scales defined by other means are supposedly in agreement with the thermodynamic scale or differ from this scale by small, known amounts.

Primarily the high temperature scale from 100°C. to about 1,500°C. has been established by the use of constant-volume nitrogen gas thermometer. If an "ideal" gas existed and were employed in the gas thermometer, the temperature scale would agree throughout this range with the thermodynamic scale. Such a gas does not exist, but the corrections which are necessary to apply to the nitrogen gas-thermometer scale to make this scale agree with that of an ideal gas are known. These corrections are small amounting to only 0.05° at 200°C., 0.3° at 600°C., and 1° at 1,200°C.

The gas thermometer is difficult and inconvenient to operate and is not employed even for precision work except as a fundamental standard instrument. It would not be feasible to calibrate all other types of pyrometer directly by comparison with the gas thermometer. Accordingly a series of so-called fixed points have been adopted, defined by the melting or boiling points of several chemical elements and compounds. The temperatures at which these materials melt or boil have been determined up to 1,500°C. by the gas thermometer. In these primary experiments a thermocouple or resistance thermometer served as a transfer instrument since it was not usually possible to place the bulb of the gas thermometer directly in the melting or boiling chemical element. Having obtained by means of the gas thermometer a number of fixed points which could be corrected to conform with the true thermodynamic temperature scale, it was found that the temperature scale defined by other forms of pyrometer agreed with the thermodynamic scale as closely as this scale could be determined by the gas thermometer, and that the precision possible in other pyrometers is far greater than that of which the gas thermometer is capable.

Thus, it has been found that the high temperature scale defined by the platinum resistance thermometer calibrated in terms of the melting point of ice, 0°C., the boiling points of water, 100°C., and of sulphur, 444.6°C., at a pressure of 760 mm. Hg.,

[1] U. S. Bureau of Standards.

represents the true thermodynamic scale (as closely as we have any means whatever of determining this agreement) up to 1,100°C., the highest temperature at which the resistance thermometer can be operated satisfactorily. If the scale defined in this manner is arbitrarily chosen as the working temperature scale, we have a means of measuring temperature far more precisely than is possible with the gas thermometer. If the boiling point of sulphur should be found at any time to differ slightly from the above accepted value, the scale defined by the platinum resistance thermometer could be altered by known small amounts to conform with the newly accepted value.

The gas thermometer has never been used at a temperature higher than 1,550°C. Above 1,500°C. the temperature scale is defined by means of the Stefan-Boltzmann or Wien-Planck radiation laws. These laws have a theoretical significance, and experimental evidence is such as to lead to the conclusion that the scales defined by these two radiation laws are in mutual agreement and that they represent the ideal thermodynamic scale.

The provisional scale adopted by the Bureau of Standards may be expressed in terms of the following fixed points. On the basis of the true thermodynamic scale these standard temperatures are known to an accuracy of possibly 0.5° at 500°C., and 3° at 1,200°C. On the basis of the platinum resistance thermometer scale defined as above, the temperatures below 1,000°C. can be determined with possibly 10 times this precision. The accuracy with which the platinum point is known on the thermodynamic scale is probably 10°C., and the accuracy of the tungsten point may be estimated as 50°C.

TABLE 1.—FREEZING OR MELTING POINTS, DEGREES CENTIGRADE

Tin	231.9	Gold	1,063
Lead	327.3	Copper	1,083
Zinc	419.4	Nickel	1,452
Antimony	630.0	Palladium	1,550
Aluminum	658.7	Platinum	1,755
Copper-silver eutectic	779.0	Alumina	2,050
Silver	960.5	Tungsten	3,260

BOILING POINTS

Naphthalene	217.96°C. + 0.058(P − 760)[1]
Benzophenone	305.9 °C. + 0.063(P − 760)[1]
Sulphur	444.6 °C. + 0.090(P − 760)[1]

The above table applies for materials of the highest possible purity. The presence of impurities will, in general, lower freezing or melting points and raise boiling points.

HIGH-TEMPERATURE THERMOMETRY

Introduction.—High-temperature thermometry deals with instruments used to measure temperature in the range 100 to 550°C. This temperature range is more or less arbitrary, although the upper limit (550°C.) is fairly definite and coincides with the approximate upper limit of use of the mercury-in-glass thermometers.[2]

In this range (100 to 550°) the domains of pyrometry and thermometry overlap somewhat, for high-temperature resistance thermometers and thermocouples generally classed as pyrometers are often used below 550°C. These instruments will not be

[1] P denotes pressure in mm. Hg.

[2] Quartz glass thermometers filled with mercury under pressure have been constructed to measure temperatures up to 750°C. They are not used in this country, however, except to a negligible extent. Reports as to their behavior have not been promising.

discussed in this section which pertains more especially to high-temperature mercurial thermometers, and indicating and recording thermometers of the vapor pressure, liquid or gas filled and bimetallic types, a brief classification of which is given in the table following:

TABLE 2.—CLASSIFICATION OF THERMOMETERS

General classification		Subdivisions
Mercury in glass (mercurial)	Laboratory or chemical	Etched or engraved stem
		Enclosed scale (einschluss)
	Industrial	
Indicating and recording	Pressure	Vapor pressure
		Liquid filled
		Gas filled
	Bimetallic	
	Graphite-metal expansion	

High-temperature Mercurial Thermometers.—In the laboratory the so-called engraved- or etched-stem thermometer is more extensively used in this country than the enclosed scale or einschluss type. Figure 1 shows characteristic types of etched-stem high-temperature laboratory thermometers. The first instrument (*a*) is the type used as a standard at the Bureau Standards in the range 300° to 530°C. The second thermometer (*b*) is a continuous-scale instrument used for making distillation tests and graduated 0° to 400°C. in 1° intervals for total immersion, the third instrument (*c*) is a 20 to 750°F. thermometer divided into 2° intervals for partial immersion and used in oil testing or other industrial work. In the use of high temperature thermometers, the ice-point graduation (0°C., 32°F.) is desirable for checking purposes, since the indications of even the best made instruments fail to remain constant on repeated heating, due to a slow contraction of the glass in the bulb. Observations made from time to time of the change of readings in ice, may be used to correct the readings over the entire scale.

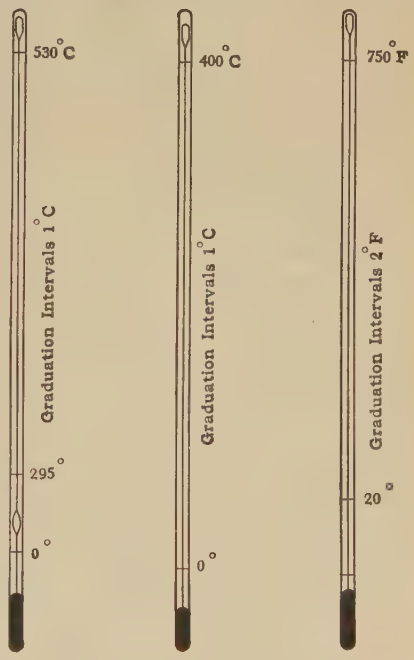


FIG. 1.—Etched stem thermometers.

If it is necessary to use a thermometer for partial immersion, that is, with the bulb and only a part of the stem heated, it is advisable to use the continuous-scale type (*b*) or (*c*) rather than the standard type (*a*), especially if the enlargement in the capillary shown between the 0° and 295° marks projects out into the air which is much cooler than the bulb. Large and uncertain errors may be introduced by using a thermometer under these conditions.

It is not necessary or advisable to graduate thermometers to be used in the range 200 to 500°C. into intervals smaller than 1° or 2°C. The undesirability except under unusual conditions of specifying, for example, six thermometers graduated in 0.1° intervals with scale range of 25° each to cover the range 200 to 350°C. instead of one instrument graduated into 1 or ½° cannot be too strongly emphasized.

These short range thermometers are seldom provided with ice point graduations for checking and for this reason in general cannot be standardized to even as high a degree of accuracy as thermometers divided into 1 or ½° intervals. Very little is to be gained therefore by attempting to read these short scale instruments to 0.05 or 0.01°C. when the maximum obtainable accuracy is from 0.1 to 0.5°C. The following table gives accuracy data for laboratory thermometers in the range 100 to 500°C., and 212 to 1,000°F.

TABLE 3

Approximate scale range	Subdivision	Probable accuracy of unstandardized thermometers	Accuracy which may be obtained if thermometer is standardized
		Centigrade Scale	
100 to 200°C.	5°	2.0°	1.0°
100 to 200°C.	2	1.0	0.5
100 to 200°C.	1	1.0	0.2
100 to 200°C.	⅕ or ½	0.5	0.1
200 to 300°C.	5°	2.0°	1.0°
200 to 300°C.	2	2.0	0.5
200 to 300°C.	½ or 1	1.0	0.2
300 to 400°C.	5°	3.0°	2.0°
300 to 400°C.	2	3.0	1.0
300 to 400°C.	1	2.0	0.5
400 to 500°C.	5°	5.0°	2.0°
400 to 500°C.	1 or 2	2.0	1.0
		Fahrenheit Scale	
212 to 400°F.	5°	2.0°	1.0°
212 to 400°F.	2	1.0	0.5
212 to 400°F.	½ or 1	1.0	0.1
400 to 600°F.	5°	2.0°	1.0°
400 to 600°F.	2	2.0	0.5
400 to 600°F.	1	1.0	0.2
600 to 800°F.	5°	3.0°	2.0°
600 to 800°F.	2	2.0	1.0
800 to 1,000°F.	5°	5.0°	2.0°
800 to 1,000°F.	2	4.0	2.0

The above table was compiled for use at the Bureau of Standards, and is only applicable to thermometers of the laboratory type used under total-immersion conditions, *i.e.*, with bulb and stem containing the mercury column at the same temperature.

The table emphasizes the necessity for obtaining and applying corrections to the readings of thermometers especially if high accuracy is desired. The maximum errors allowed in the table represent the probable magnitude of the error that may be expected of the best grades of thermometers. This degree of accuracy cannot always be expected of cheaper grades of thermometers, or of partial-immersion thermometers.

Industrial Thermometers.—The use of the chemical or laboratory type of mercurial thermometer is restricted as the name implies to the laboratory since unprotected glass thermometers are much too frail to withstand the rough usage of the plant and shop, and are comparatively difficult to read as well.

To meet the requirement of the plant there has been developed what is generally known as the industrial type of mercurial thermometer. As shown in Fig. 2, it is characterized by a heavy metal back and protecting tube for the bulb, large and distinct figures and graduation marks, and threaded connections for attaching the instrument firmly and quickly to some part of the apparatus.

This same general design of instruments is used for many different operations, covering ranges of temperature from −30° to 1,000°F. These instruments necessarily must be graduated and standardized for the condition of use, that is for partial immersion of the mercury column.

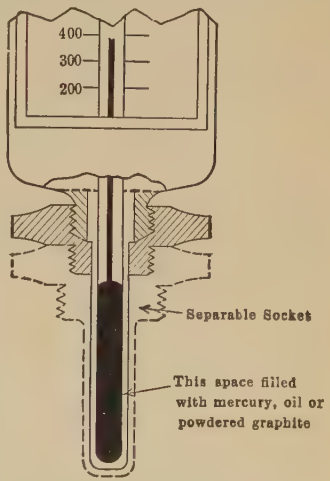


Fig. 2.—Industrial thermometer.

It will be noted from the cross-section view of the bulb and protecting case of the industrial thermometer, that first, the bulb does not come into direct contact with the substance the temperature of which is to be measured, and secondly the bulb is surrounded by large metal parts which extend into a region which is at a different temperature from that of the bulb.

These two peculiarities in the construction of an industrial thermometer must be taken into consideration in their use. Since the bulb is not in direct contact with the heated substance the time that it requires to take up changes in temperatures is greater than that necessary for a bare bulb. This "lag" as it is called can be reduced by filling the space between the bulb and the outer wall of the casing with a good conducting medium. The most satisfactory substance to use for temperatures below 200°C. is mercury. Experiments made at the Bureau have demonstrated that the lag of an instrument read in steam at 212°F. with mercury surrounding the bulb, was ½ to ⅓ as great as when powdered graphite, oil, or air was used. Obviously mercury cannot be used at the higher temperatures, and heavy oils are usually substituted.

This "lag," however, may not be as great a source of error as the actual variance which may be noted in the readings of these thermometers when used to measure the same temperature, but under varying conditions as regards construction of the protecting case surrounding the bulb, the material into which the bulb is immersed, *i.e.*, whether liquid, vapor or gas, the rate of flow of the material past the bulb, and the exterior conditions surrounding the protecting stem.

These variations can be understood by examination of the construction of the thermometer and an analysis of the means whereby the heat is either conducted to the

bulb of the instrument or is lost by conduction to the air by way of the heavy metal parts. There are no published reliable investigations giving data in regard to the magnitude of the influence of all the factors noted above.

If industrial thermometers are used to indicate only *changes* in the temperature of a process, and not to give actual temperatures, it may not be necessary to consider the various sources of error enumerated, but it should be emphasized that these instruments cannot be depended upon to give reliable indications of true temperatures, unless they are calibrated or standardized under the same condition as that of use, and after being put in use these conditions are not changed.

Emergent Stem Correction for Mercury-in-glass Thermometers.—A large proportion of the thermometers now sold are graduated and standardized for the condition of total immersion, that is, for the condition that the bulb and all the portions of the stem which contain mercury, are heated to a uniform temperature.

The thermometers are, however, rarely used under this condition, but in nearly all cases are used at partial immersion, the bulb and a short length of the stem being exposed to the temperature to be measured, while the remainder of the stem projects above the bath into the (usually) cooler air of the room. In the following it will be assumed that the thermometer is in a heated bath. In this case, the emergent mercury column is cooler than the bath, consequently the thermometer reads too low. The amount, in degrees, by which it reads too low, may be calculated from the formula,

$$S = an(T - t)$$

where S is the correction to be applied to the reading, a is the relative expansion of mercury in glass (0.00016 for centigrade mercurial thermometers and 0.00009 for Fahrenheit thermometers), n is the number of degrees of the mercury column emergent from the bath, T is the temperature of the bulb or bath, and t is the mean temperature of the emergent column. It may be noted that if t is a higher temperature than T, the sign of the correction will be negative, so that, if due attention is given to the sign, the formula holds whether the bath is warmer or colder than the emergent stem.

TABLE 4.—STEM CORRECTION DATA FOR CLEVELAND OPEN-CUP FLASH AND FIRE-POINT TESTER

Thermometer range −20 to 760°F. in 2° intervals, length about 15 in.

Temperature of bulb, degrees Fahrenheit	Degrees of mercury column emergent, degrees Fahrenheit	Mean temperature of emergent mercury column, degrees Fahrenheit	Stem correction, degrees Fahrenheit
208	200	173	0.5
308	300	174	3.5
408	400	169	8.5
508	500	172	15.0
608	600	180	23.0

Of the factors in the formula, the value of a has been given, n can be observed while T is (except for the error due to the emergent stem) equal to the reading of the thermometer. The value of t is the most difficult to determine, but an approx-

imate value can be obtained by the use of an auxilliary thermometer, the bulb of which is placed about three-fourths of the way down the exposed mercury thread.

Tables 4, 5 and 5A show the magnitude of the emergent stem correction in some typical cases.

TABLE 5.—STEM CORRECTION DATA FOR DISTILLATION FLASK USED IN DISTILLATION OF CREOSOTE OIL[1]

Dimensions of flask: Diameter bulb 8 cm., diameter neck 2 cm., height neck 15 cm., distance bottom of outlet tube to top 8 cm. Thermometer range 0 to 400°C. in 1° intervals, length 40 cm.

Temperature of bulb, degrees Centigrade	Degrees of mercury column emergent	Mean temperature of mercury column	Stem correction
208	200	59	4.7
258	250	105	6.0
308	300	87	10.5
358	350	79	15.5

TABLE 5A.—STEM CORRECTION DATA FOR PENSKY-MARTIN FLASH-POINT APPARATUS

Thermometer used up to 150°C., range 40 to 160°C. in 1° intervals, scale length 9.5 cm. Thermometer used from 200 to 300°, range 200 to 360° in 1° intervals, scale length 12 cm

Temperature of bulb, degrees Centigrade	Degrees of mercury column emergent, degrees Centigrade	Mean temperature of emergent mercury column, degrees Centigrade	Stem correction
50	30	35	0.1
100	80	45	0.7
150	130	55	2.0
200	10	75	0.2
250	60	85	1.6
300	110	100	3.5

Partial Immersion Thermometers.—As a means of avoiding the necessity for applying the correction for the emergent stem, it has become customary to use partial immersion thermometers which are graduated to read correctly when immersed to a definite depth, say three inches, in a bath. Such thermometers should be, and usually are, marked to show the proper depth of immersion, and many of them have a line engraved around the stem to show this depth of immersion. While the accuracy attainable with a partial immersion thermometer is not so high as *can be* attained with a total immersion thermometer *under favorable conditions*, it is true that these favorable conditions are not often realized, and therefore in the great majority of cases, the partial immersion thermometer is to be preferred. It is probable that the partial immersion thermometer will be extensively used in this country.

[1] See paper by Wilhelm, Bureau of Standards *Tech. Paper* No. 49.

In general, it may be said that in all cases where the application of stem corrections is neglected, which includes a vast majority of ordinary routine laboratory temperature measurements, more accurate temperature measurements would be attained by the use of thermometers graduated as partial immersion thermometers. The same statement would apply for measurements at the higher temperatures (above 200°C. or thereabouts), even if stem corrections are applied, when the ordinary method of estimating average stem temperature is used instead of the more accurate capillary thermometer method. At the lower temperatures, on the other hand, a slight advantage rests with the total immersion thermometer, if the stem correction is determined and applied in the usual manner, *i.e.*, by the intelligent use of an auxiliary thermometer to determine the average stem temperature. Thermometers graduated in intervals smaller than 0.5°C should not, in general, be graduated as partial immersion thermometers, if the accuracy of which they are capable is desired, unless such finer graduation be deemed of sufficient importance solely from the standpoint of convenience in reading.

Other Sources of Error in High-temperature Mercurial Thermometers.—Aside from the error that may be introduced in the use of high temperature mercurial thermometers by a failure to observe the proper condition of immersion, there are two common sources of error that can be attributed to actual defects in the manufacture of the instrument. These are first, insufficient pressure above the mercury to prevent a breaking of the mercury column, or evaporation at the surface of the mercury, and secondly, improper or insufficient annealing for use at high temperatures. This latter defect may result in a rise of the reading with continued heating amounting to as much as 20°C. in extreme cases.

In discussing the question of filling under pressure it may be noted that mercury boils at approximately 357°C., and evaporates fairly rapidly at much lower temperatures. Experiments have shown that even for use above 100°C. the filling under pressure of that part of the capillary above the column with an inert gas such as nitrogen is desirable. The amount of pressure that must exist above the surface of the mercury to prevent evaporation or breaking of the mercury column varies with the temperature and the construction of the upper portion of the stem. This pressure may be anywhere from 2 to 20 atmospheres.

The failure to fill high-temperature thermometers under the proper pressure is often the cause of large errors. This defect can be detected sometimes by the broken appearance of the mercury column, and by drops of mercury that have distilled from the top, but often the column breaks in a part of the stem not visible, and the defect is not detected.

Improper annealing can be detected only by the checking of the indications of the instruments from time to time. Laboratories usually have facilities for checking the reading either in melting ice or in steam or perhaps comparing the thermometers from time to time with a thermometer known to give reliable readings. Industrial thermometers should also be tested for this source of error, although annealing changes due to heating are not liable to be appreciable below 300°C. for this type.

Indicating and Recording Thermometers.—The term "indicating" is usually employed to designate thermometers of the dial-and-pointer type, rather than those constructed of mercury and glass. Indicating thermometers may or may not be distance-reading, *i.e.*, so constructed as to allow the indicator to be placed at some distance from the bulb. A distance-reading indicating thermometer is

shown in Fig. 3. A recording thermometer employs a mechanism for making a continuous record of temperatures on a suitable chart.

Indicating and recording thermometers may be divided into three general classes, electrical, pressure, and bimetallic. Electrical thermometers will be discussed in later sub-sections ("Thermocouples," "Resistance Thermometers").

Pressure thermometers comprise a bulb containing a liquid or gas or both connected by means of capillary tubing to some form of pressure gage.

Bimetallic thermometers utilize the turning force produced when a strip consisting of two metals having different coefficients of expansion and brazed to each other, is heated. Graphite-metal thermometers indicate temperatures as a result of the relatively large difference in thermal expansion of the two substances. Although of simple design, the accuracy and adaptability of these types of instrument has not been sufficiently investigated to allow a more detailed discussion of them here.

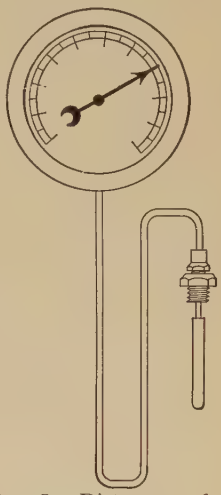


FIG. 3.—Distance-reading thermometer.

Pressure thermometers may be sub-divided into vapor-pressure, liquid-filled and gas-filled instruments. From outward appearance vapor-pressure thermometers may be distinguished from liquid- or gas-filled instruments by the form of the temperature scales. Vapor-pressure thermometers have an unequally divided scale, the length of the intervals for a given number of degrees increasing with increasing temperature. Liquid- and gas-filled instruments have equally divided scales. The bulbs, capillary tubing and form of pressure gage may be identical in outward appearance.

The pressure under which a vapor-pressure thermometer operates is a function of the temperature of the bulb only. The liquid selected to cover a given temperature range should have sufficient pressure at the lower temperatures to insure a readable temperature scale in this region. The lower the initial pressure the more constricted the lower part of the temperature scale will be as compared with the upper part of the scale where the range of pressure for a given change in temperature is much greater. The maximum temperature to which the liquid used in a vapor-pressure thermometer will be heated should not exceed its critical temperature. In general the temperature range for a given liquid is included between its boiling point at atmospheric pressure and its critical temperature.

The initial pressure and the pressure range of a liquid- or gas-filled thermometer will vary according to the strength of the gage used and the temperature range. Since the increase in pressure with temperature is approximately the same over the entire scale, the temperature scale is evenly divided. The pressures in liquid- or gas-filled thermometers are made relatively high to insure a rugged gage mechanism. The following table gives some characteristics of vapor-pressure and liquid- or gas-filled industrial thermometers.

The action of the vapor-pressure thermometer depends upon the fact that the pressure inside the thermometer is determined solely by the temperature of the free surface of the liquid. It follows therefore that the thermometer must be so constructed that one free surface is always in the bulb. If this condition is fulfilled the reading of the instrument will not be sensibly affected by changes in the temperature of the gage and capillary.

TABLE 6.—CHARACTERISTICS OF PRESSURE THERMOMETERS

Designation	Material available for filling	Approximate temperature range, degrees Centigrade	Scale
Vapor pressure	Ethyl ether	40 to 190	Unevenly divided
	Ethyl alcohol	80 to 240	Unevenly divided
	Benzene	80 to 280	Unevenly divided
	Aniline	200 to 425	Unevenly divided
Liquid filled	Alcohol	up to 200	Evenly divided
	Mercury	up to 550	Evenly divided
Gas filled	Nitrogen	up to 425	Evenly divided

This is a decided advantage over other types, if the connecting tubing and gage are both to be subjected to considerable changes in temperature. On the other hand, if the vapor-pressure thermometer is not filled properly, *i.e.*, the proportion of liquid is too great or too small as compared with the volume of the bulb, capillary and gage, large and uncertain errors may be introduced. As an example take the condition met with in the use of high-temperature vapor-pressure thermometers. The bulb is usually much hotter than the capillary which is filled or nearly filled with vapor so that the liquid is condensed in the capillary and there should be sufficient liquid in the system to entirely fill the capillary and gage and partly fill the bulb. If the liquid only partly fills the capillary, and there is no liquid in the bulb, the capillary will contain the free surface of the liquid and the temperature indicated will be that of the portion of the capillary containing the free surface of the liquid as shown in Fig. 4. This temperature, which may be several hundred degrees lower than that of the bulb, will be indicated instead of the true temperature of the bulb.

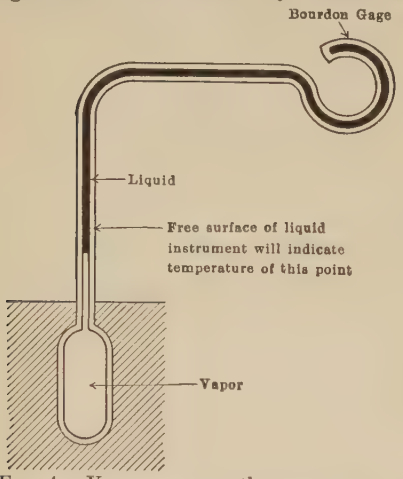


FIG. 4.—Vapor-pressure thermometer.

Gas- and liquid-filled thermometers operate on the principle of thermal expansion. They are entirely filled with either the liquid or the gas. The expansion of the liquid or gas in the bulb is transmitted through capillary tubing to the pressure gage. These instruments are subject to error if the gage and capillary are heated or cooled to temperatures differing from those under which they were calibrated. This error may be made negligible in many instances by reducing the volume of the capillary and gage as compared with that of the bulb, or by using a compensator in the head. Such a compensator, however, will not eliminate errors due to the heating or cooling of the capillary alone.

On account of the mechanical construction of pressure thermometers, both as regards the mechanism for indicating and recording and the necessity for using comparatively large and heavy bulbs, the accuracy that can be secured with this type

of thermometer is lower than that obtained with mercury-in-glass thermometers. As with industrial thermometers, the conditions under which the instruments are originally calibrated must agree with those of use if errors are to be avoided. The greatest error is introduced, perhaps, when such instruments are calibrated in a liquid, and are used to measure the temperature of a gas.

THERMOELECTRIC PYROMETRY

Thermocouples.—Seebeck discovered in 1821 that if in a closed circuit of two metals the two junctions of the metals are at different temperatures, an electric current will flow in the circuit. For example, if the ends of an iron and of a copper wire are fused together and one of the junctions is heated (within certain temperature limits) positive current will flow in the direction copper to iron at the hot junction or iron to copper at the cold junction. Two causes contribute to the production of the electromotive force which actuates this current: An electromotive force exists between two different metals placed in contact, the magnitude of which depends upon the temperature and upon the metals used; also if a wire of homogeneous material is heated at one end, an electromotive force is developed between the hot and cold ends of the wire, the magnitude of which depends upon the metal and upon the differences in temperature of the ends. These two electromotive forces are known as the Peltier electromotive force and the Thomson electromotive force respectively. The total electromotive force acting in the copper-iron circuit described above is the sum of the Peltier electromotive force at each junction and the Thomson electromotive force over each wire, consideration being given, of course, to the algebraic signs of these four electromotive forces. The total electromotive force acting in a circuit of two dissimilar metals thus depends upon the temperatures of the two junctions. If the temperature of one junction is fixed at that of the room or of melting ice, etc., the temperature of the other junction can be determined by measuring the electromotive force developed in the circuit.

This is the basic principle of thermoelectric pyrometry. The electromotive forces developed by thermocouples are small, usually a few thousandths of a volt. To measure such small electromotive forces special types of sensitive voltmeters (millivoltmeters) or indicators are required. For any particular type of couple these instruments may be graduated to read temperature directly instead of electromotive force.

A simple thermoelectric pyrometer consists of three parts:

(*a*) The thermocouple of two different metals or alloys having a fused junction (the hot junction) which is inserted into the furnace and the cold junctions which protrude from the furnace and are maintained at some fixed temperature such as that of the room or of melting ice;

(*b*) Two lead wires, usually of copper, running from the cold junctions of the thermocouple to the indicator;

(*c*) The indicator which may be a millivoltmeter, a potentiometer or a special type of instrument embodying both of these principles, and may be graduated to read either e.m.f., temperature or both.

Metals Used for Thermocouples.—Although any two dissimilar metals might be employed for a thermocouple, certain combinations are unsatisfactory because of the very small electromotive forces which can be developed and because of the fact that with some combinations the electromotive force may

first increase, then decrease, become zero and change sign as the temperature increases. Obvious desirable properties for a thermocouple are: (1) Capability for resisting corrosion and oxidation. (2) The development of relatively large electromotive forces. (3) A temperature-electromotive force relation such that the electromotive force increases continuously with increasing temperature over the temperature range employed.

For general work at higher temperatures, several different types of couples are employed in this country. Up to 360°C. for extreme precision or to 500°C. for a precision of 5 to 10°C. the couple may consist of one wire of copper and the other wire of constantan. Iron-constantan or nichrome-constantan may be employed for technical processes below 900°C. For operation below 1,100°C. special patented alloys of chromium and nickel and of aluminum and nickel, "chromel-alumel" or "nichrome-alumel" are very satisfactory even for continuous service. For the temperature range 300 to 1,500°C. the Le Chatelier couple should be employed. This couple consists of one wire of platinum and the other wire an alloy containing 90 per cent platinum and 10 per cent rhodium. Other alloys and metals may be employed for special work but the above combinations are sufficient for almost all technical processes carried on in the temperature range 0 to 1,500°C. No satisfactory couple has been developed for operation much above 1,500°C.

Constancy of Calibration.—A further essential property of a thermocouple is constancy of calibration. Alterations in the calibration may be due to various causes. Inhomogeneity may develop on account of contamination by fumes from the furnace, metallic vapors, by oxidation, etc. Contamination may be usually prevented by the use of suitable protecting tubes surrounding the couple; and the effect of contamination may be minimized by using wire of large cross-section. Different furnace conditions and different types of couples require different methods of protection against contamination. Each process must be considered individually. For example a platinum couple is usually protected against contamination by means of refractory porcelain tubes, but if the atmosphere surrounding the platinum is reducing, the use of porcelain may do more harm than good. In this case, the reducing atmosphere changes the silica of the porcelain into silicon which readily attacks the platinum. The electromotive force of some couples gradually decrease with use. The platinum-platinum 90 per cent, iridium 10 per cent, couple has not proven very satisfactory for this reason, although it develops a much larger electromotive force than the platinum-rhodium thermocouple. The iridium gradually distills from the alloy wire, especially above 1,200°C., requiring frequent recalibration of the couple.

Reproducibility of Couples.—When thermocouples are employed in the laboratory for scientific purposes, although desirable, it is not of serious importance that the calibration or temperature-electromotive force relation of couples of the same type be exactly similar. However, in the industrial plant this question of reproducibility is of considerable moment. The indicating instruments are usually graduated in degrees of temperature and the graduation applies for one definite temperature-electromotive force relation only. If the temperature electromotive force relations of various couples of the same type are not similar, corrections must be applied to the readings of the indicator, and these corrections will be different for each couple. When several couples are operated with one indicator and when the process is such as to require a frequent renewal of couples, the applying of these corrections becomes very troublesome.

For extreme precision it is always necessary to apply such corrections, but for most industrial processes, thermocouples which are sufficiently reproducible can be secured, so that the corrections are negligibly small and may be omitted. Thus the temperature-electromotive force relations of different homogeneous and uncontaminated chromel- alumel couples should not vary in general by more than 5 or 10°C. and of platinum-platinum rhodium couples by more than 2 or 3°C. There are few industrial processes carried out at high temperatures under such accurate temperature control that the variations in the calibration of the different new platinum-platinum rhodium thermocouples of the same type warrant consideration. The variations in the calibration of different base-metal couples are frequently corrected by use of series or shunt resistance, but most methods so far devised are rather unsatisfactory and some of the compensation methods after continued use may develop larger errors than those arising from the irreproducibility of the couple. The above statements in regard to reproductibility of different couples of the same type apply only for new couples. After a couple has been used for some time, especially a base-metal couple, or has become contaminated in any manner, the calibration may change considerably. All thermocouples should be checked frequently, as installed, to determine whether or not the calibration has altered.

Mounted Thermocouples.—The question of properly mounting and protecting a thermocouple is of great importance. The type of protection necessary depends upon the particular industrial process for which a couple is employed. In the laboratory if it be desired to measure the temperature of a perfectly clean platinum-wound porcelain furnace containing no material such as iron, etc. which could contaminate platinum, a rare-metal couple may be used without any protection whatever. For most laboratory experiments, however, a rare-metal couple requires protection. The usual rare-metal couple consists of wires 0.5 mm. or preferably 0.6 mm. in diameter and from 50 to 125 cm. in length. Wires as small as 0.1 mm. and even less are frequently used for special research work. One or both of the wires are insulated by threading them through small porcelain or quartz tubes.

In measuring temperatures below about 1,400°C., two-hole porcelain tubes are very useful for insulating both wires, but for higher temperatures separate tubes should be used. The hot junction of the couple is made by fusing the two wires in an arc or oxygen-illuminating gas flame. The couple and insulating tubes are inserted in a small outside protecting tube of porcelain glazed on the outside only, or of fused silica, hemispherically closed at one end. On the open end of the protecting tube may be mounted the head of the couple which serves as a handle and as a support for rigidly holding the wires of the couple. The couple wires frequently extend beyond the head of the couple so that their ends may be maintained at some controlled cold junction temperature. Usually the wires terminate in binding posts on the couple head, in which case the cold-junction temperature may be controlled by water jackets, or may be allowed to remain that of the surroundings, or the couple may be fitted with one of the various devices discussed later for the elimination of cold-junction errors.

Base-metal couples for laboratory use may be constructed in much the same manner described above and may be made of wire as small as No. 20 or of very much smaller wire for certain types of research at lower temperatures. Small wires, however, are readily oxidized completely at high temperatures so that for continuous operation in industrial installations the couples are constructed of No. 8 or No. 6 wire or of still larger wire when there is danger of contamination. The hot junction is fused as in the case of rare-metal couples, and usually the two wires are twisted

for a few turns at the hot junction in order to give greater mechanical strength to the joint. The two wires are insulated by fireclay insulating tubes, or by asbestos sleeving or cord, and are connected to a suitable couple head forming the cold junction. For severe use it is necessary to encase the couple in a protecting tube of steel, chromel, porcelain, fireclay, etc. A different form of base-metal couple, known under the trade name of "pyod" consists of an outer tube of iron and an inner wire or rod of constantan. The two are fused at one end into a neat joint forming the hot junction and are insulated from each other up to the head or cold junction. The couple is thus mechanically stronger than couples formed of two wires and when used without an additional protecting tube, is somewhat less liable to contamination than the bare two-wire couple of the same type, since the constantan, which is probably more susceptible to contamination than the iron, is protected by the iron tube of the pyod. However, couples of this type must be further protected by outer tubes if subjected to severe furnace conditions. Couples of various shapes may be constructed for special purposes. Thus frequently it is desirable to make a right-angle bend in a couple. Usually a mounted couple will stand bending into various shapes but before bending one should examine the couple and temporarily remove porcelain tubes or insulators liable to be broken by the process.

The indicating instruments which are connected to the thermocouple are of three general types, those operating upon the galvanometric principle, as an ordinary voltmeter; those operating upon the potentiometric principle; and those operating upon a combination of these two principles. The first two types of instrument have been made automatically recording, as will be discussed elsewhere.

Galvanometer Method.—Galvanometers for measuring the electromotive force developed by a thermocouple usually operate on the d'Arsonval principle, having a moving coil mounted between the poles of a permanent magnet. The deflection of the pointer or boom attached to the coil is proportional to the thermoelectric current passing through the coil, and the current is proportional to the electromotive force developed by the couple. Different methods for mounting the coil are employed. The coil may be suspended both above and below by phosphor-bronze suspensions and although many foreign instruments of this type have proven delicate, in one of the latest forms of American instrument using this double suspension the galvanometer may be subjected to severe handling without any injury.

The scale of the instrument may be graduated to read electromotive force or temperature. By use of a series resistance mounted inside the galvanometer case and an extra binding-post terminal, two scale ranges may be obtained, one for base-metal couples of a certain type and the other for the rare-metal LeChatelier couple. For precision work, the scale should have a narrow mirror extending throughout its length for use in eliminating the error in reading due to parallax. While obtaining a reading the observer should use one eye only and he should move his head to the position where the pointer and its reflected image in the mirror coincide.

Indicators may be obtained in either the switchboard type or portable type. The former is desirable for permanent installations. Many switchboard instruments have a scale ruled on ground glass and illuminated from the rear by an electric lamp. The black-enameled graduations upon the scale thus appear distinct against a bright background. Paper scales illuminated from the front are usually employed. In general high precision is not required of an instrument of the switchboard type so that the graduations may be coarse and readable at some distance. However, it is possible to obtain highly accurate switchboard indicators in which case the scale

should have a mirror to permit readings without parallax. If the instrument is mounted in a dusty room it should be protected by an outside case of steel or wood having a glass front, as the presence of dirt filtering into the interior of the galvanometer may seriously affect its calibration. Both the wall type and the portable indicator, especially the latter, must be sufficiently robust to withstand reasonably severe usage.

Resistance of Indicating Instruments.—When operated at the highest safe working temperatures most base-metal couples develop a maximum electromotive force of less than 50 to 70 millivolts and the LeChatelier couple an electromotive force of about 16 millivolts. In order to measure such small electromotive forces accurately a very sensitive indicator or millivoltmeter is required. On the other hand the instrument must be robust and able to withstand rough handling to which it is necessarily more or less subjected. These conditions of mechanical robustness and of high accuracy as a pyrometer indicator are difficult to satisfy.

The deflection of the pointer of the millivoltmeter is approximately proportional to the current flowing through the coil so that if the resistance of the total thermoelectric circuit is very low, relatively large currents are obtained, with a resulting high torque or turning moment on the movable coil. Thus when the current is high the construction of the indicator may be made very robust. Relatively strong springs, against which the turning moment of the coil carrying the current is balanced, may be employed and less attention need be given to the friction of the pivots in their bearings. The torque may be made high without greatly increasing the resistance of the circuit by using a large number of turns of copper wire in the coil. Thus it is not especially difficult to construct a sufficiently sensitive millivoltmeter of low resistance. The objection, however, from the pyrometric standpoint, to such an instrument used as a simple galvanometer has led to the development of indicators of considerable resistance. These are made by placing a high resistance of zero temperature coefficient, such as manganin, in series with the coil, and by increasing the number of turns on the moving coil to compensate for the decrease in sensitivity arising from the increased resistance. The so-called swamping resistance of zero temperature coefficient may be so proportioned relative to the copper that, due account being taken of the temperature coefficient of elasticity of the springs, the instrument as a whole possesses a negligible temperature coefficient. The use of high resistance greatly diminishes the magnitude of the current flowing through the coil and thus decreases the torque producing deflection. Hence attention must be given to the elimination of bearing friction as far as possible. Thus the instrument is necessarily more delicate than an indicator of low resistance and of the same type. The advantage of an instrument having a high resistance is that changes in the line resistance of comparatively large magnitude do not greatly affect the readings, as shown by the following table, the resistances of the indicators being 300 and 10 ohms respectively and the *normal* line resistance 2 ohms.

TABLE 7.—ERROR DUE TO CHANGE IN LINE RESISTANCE

Line resistance, ohms	Per cent error in indicator reading		Error in degrees at 1,000°C.	
	300 ohms	10 ohms	300 ohms	10 ohms
1	+0.33	+ 9.1	+3.3	+ 91
2	±0.00	± 0.0	±0.0	± 0
3	−0.33	− 7.7	−3.3	− 77
5	−0.66	−14.3	−6.6	−143

Thus when both instruments read correctly for an external resistance of 2 ohms if the external resistance is increased by an ohm, the indicator of low resistance is in error by 7.7 per cent or about 77°C. at 1,000°C., while the instrument of high resistance still reads practically correct. Corrections such as the above emphasize the importance of using a galvanometer having a resistance of 300 ohms or more. In actual operation the line resistance may change by several ohms on account of bad contacts and the deterioration of the thermocouple. The error due to variable line resistance is eliminated in the Harrison-Foote compensated indicator described later, even though the resistance of the indicator is low.

Use of Portable Test Set.—On account of the errors which may be introduced in the reading of an uncompensated, simple galvanometer when the resistance of the line or couple varies, it is important to have some means for measuring this resistance occasionally. This is, of course, especially necessary when the resistance of the galvanometer is low. Every plant maintaining a large thermocouple installation with simple galvanometric indicators should have a portable wheatstone bridge or "test" set for this purpose. A simple and inexpensive instrument is made by the Leeds & Northrup Co.

In measuring the resistance of the line care must be taken, if two indicators or an indicator and recorder are operated in parallel on the same couple, to see that both instruments are disconnected from the circuit during the measurement of the resistance. Otherwise the instrument left connected acts as a shunt upon the line. If the resistance of the line and couple is found to be much higher than that for which the indicator was designed, short circuit the line at the cold junction and determine whether the fault is in the couple or the line. If in the former it usually means that the couple is broken or requires replacement. By making occasional observations of this kind serious faults may be detected long before they would be suspected from the resulting low values in the indicated temperatures.

Galvanometer with Variable Series Resistance.—As shown above, galvanometers, especially those of low resistance, are usually calibrated to read correctly for a definite line resistance. Suppose an indicator is desired for operation on a line of which the resistance changes from practically zero to 10 ohms. The instrument is calibrated to read correctly for a line resistance of 10 ohms. Located in the case of the galvanometer and in series with the line is a variable resistance from zero to 10 ohms. As the resistance of the line increases the variable resistance is decreased by manual adjustment so that the sum of the line resistance and the variable series resistance always equals 10 ohms. The dial of the variable resistance is graduated to read the amount of resistance cut out of the circuit. Hence, it should be set at the resistance of the line and couple.

This value may be determined by means of a test set as discussed above. Thus the method, which has been employed by Siemens & Halske and by Weston, is of great value for precision work with a galvanometric indicator. The principal objection to it, which applies to all galvanometric indicators so far described when used for accurate measurements is the necessity of measuring the resistance of the line and couple, this latter measurement requiring the use of a test set or similar device. This objection is eliminated in the following instrument, and other desirable features have been added.

Harrison-Foote Compensated Indicator.—This instrument manufactured by the Brown Instrument Co. is illustrated in simple form by Fig. 5. The circuit *CDGF* is an ordinary millivoltmeter in which *G* represents the moving coil.

In series with this is an adjustable rheostat CB. The maximum value r_5 of this resistance is chosen equal to the maximum value of the resistance of the line and couple likely to occur in practice. A convenient value is 15 ohms. With the slide of the rheostat set for the maximum resistance, $r_1 = r_5$ the instrument is calibrated in terms of the potential drop across AH. Hence when the instrument is connected through the line of resistance L to the couple of resistance T

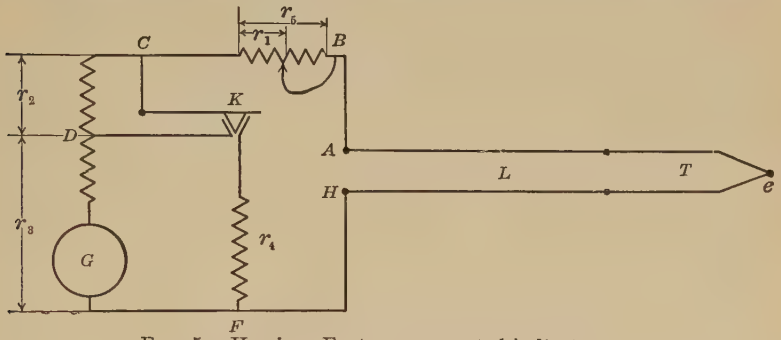


FIG. 5.—Harrison-Foote compensated indicator.

the rheostat CB must be adjusted until $r_1 + L + T = r_5$. When this adjustment is made the scale reading gives correctly the electromotive force e of the couple, or the correct temperature if the scale is graduated in degrees.

This adjustment is made in the following manner. By depressing the key K, a portion of the galvanometer resistance r_2 is short circuited and the rest of the resistance r_3 containing the moving coil, is shunted by a resistance r_4. If e' represents the potential drop across DF when the key is open and e'' represents the potential drop when the key is closed, we obtain:

$$e' = \frac{e r_3}{L + T + r_1 + r_2 + r_3} \tag{1}$$

$$e'' = \frac{e\ r_3\ r_4}{(L + T + r_1)\ (r_3 + r_4) + r_3 r_4} \tag{2}$$

If r_1 is so adjusted that these two potential-drops and hence the deflections of the indicator are the same, we have on equating (1) and (2):

$$L + T + r_1 = \frac{r_2 r_4}{r_3} = \text{a constant.} \tag{3}$$

If now in the construction of the instrument, $r_2 r_4 / r_3$ is made equal to r_5, the above setting makes $L + T + r_1 = r_5$, the adjustment required in order that the reading of the scale gives the true electromotive force of the couple.

The ease with which the proper setting can be obtained is greatly improved by making r_3/r_4 equal to from 5 to 10. Suppose r_3/r_4 is made equal to 9. Then if r_1 is improperly adjusted and the instrument reads in error by say δe when the key K is open, on depressing the key the reading is changed by $9.\delta e$. If now r_1 is readjusted with the key depressed until the reading takes its initial value, the error with the key open is reduced to $\delta e/10$. The process for operating the instrument is accordingly as follows:

1. Read the instrument with the key open.
2. Close the key and adjust the rheostat r_1 until the instrument reads approximately the same as in 1.
3. Repeat 1 and 2 if necessary.

When $r_3/r_4 = 9$ it is rarely necessary to make a second adjustment. In position 1

the instrument acts as an ordinary galvanometer. The single setting in position 2 reduces the error in the ordinary galvanometer by the factor $\frac{1}{10}$, which is usually sufficient. The adjustment for the proper external resistance, if desired, can be made with 10 times the precision necessary. Thus if the galvanometer can be read to $\frac{1}{10}$ of a scale division, the line resistance can be set for an error less than $\frac{1}{100}$ of a scale division, which is at least 10 times the accuracy possible with any indicating instrument. Hence the factor of variable line resistance which may give rise to very serious errors is easily and accurately controlled by a simple mechanical adjustment.

The device is readily applicable to multiple installations of different line resistance. For multiple-point recorders and indicators as many resistances BC may be employed as there are couples. These may be inexpensive rheostats having a resistance of approximately 15 ohms each located in each couple line between the couple and the selective switch. These rheostats may be adjusted in the manner described whenever convenient or necessary. The following illustrates a suitable proportioning of resistance for a 300 ohm indicator.

$$r_2 = 135 \text{ ohms}$$
$$r_3 = 150 \text{ ohms}$$
$$r_4 = 15\tfrac{9}{10} \text{ ohms}$$
$$r_5 = 15 \text{ ohms}$$
$$r_2 + r_3 + r_5 = 300 \text{ ohms}$$

If the simple indicator has the proper ratio of manganin to copper its temperature coefficient is practically zero. In that case the shunt r_4 should have the same manganin to copper ratio as the portion of the galvanometer resistance comprised by r_3. The complete instrument will have accordingly a zero temperature coefficient. If

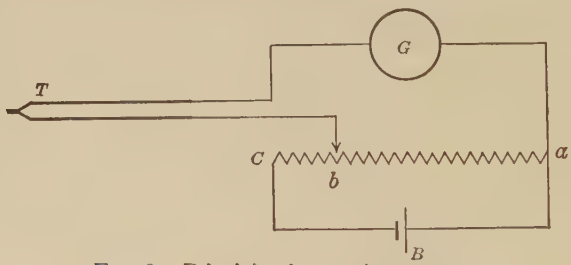


FIG. 6.—Principle of potentiometer.

the simple indicator does not have a zero temperature coefficient it is possible by increasing the proportion of manganin in r_4 to compensate for the temperature coefficient of the resistance r_3 so that the complete instrument has a zero temperature coefficient.

Potentiometer Method.—The most accurate method for measuring the electromotive force of a thermocouple is by use of a potentiometer. The fundamental principle of this instrument is illustrated by Fig. 6. A constant current from the battery B flows through the slide wire resistance abc. One wire of the couple T is connected to the movable contact b and the other wire, in series with a sensitive galvanometer, is connected to a. The contact b is moved until the galvanometer reads zero thus showing that no current is flowing through the thermocouple circuit. When this balance or zero setting is made, the true electromotive force of the couple is equal to the potential drop across ab. This is obtained from Ohm's law, $e = ir$, where i is the current flowing through the resistance $r = ab$.

If the current is always adjusted to a definite value the slide wire may be graduated to read millivolts or temperature directly. Various devices are employed for adjusting the current to the proper value. An ammeter will serve the purpose but the usual method is to obtain a preliminary setting by replacing the thermocouple by a constant known source of electromotive force such as a standard cell. The galvanometer G is always used as a zero instrument in a strictly potentiometric circuit. Hence, it requires no calibrated scale and no attention need be given to the constancy of its sensitivity provided it is always sufficiently sensitive to serve its purpose. These requirements are easily met, and the entire potentiometer, galvanometer, battery, standard cell, slide wires, etc., as constructed are mounted in a case not much larger than that of a millivoltmeter. The instrument is as mechanically robust as many indicators operating on the galvanometric principle.

There are several important advantages in the potentiometer method. The electromotive force or temperature scale is easily made very open thus permitting accurate readings. Scale lengths from 15 cm. to a meter and more are employed. The calibration of the scale is in no way dependent upon the constancy of magnets, springs, or jewel settings nor upon the level of the instrument. From the pyrometric standpoint, however, the greatest advantage is the complete elimination of any error due to changes in the resistance of the couple or of the lead wires. No matter what resistance is inserted in the thermocouple circuit, the reading of the potentiometer still gives the true electromotive force of the couple. The objections to the potentiometer are slightly greater initial cost and the fact that usually a manual adjustment must be made to obtain a setting. In the potentiometric recording instrument however all the various manipulations may be performed mechanically, even to the balancing against the standard cell, as in the recording potentiometer manufactured by Leeds & Northrup Co.

Semi-potentiometer Method.—It is possible by means of a single galvanometer or millivoltmeter to measure the electromotive force of a couple potentiometrically. Thus in Fig. 6 by using a shunted galvanometer first in the main circuit $abcB$, as an ammeter, the initial setting for a standard current is obtained. Then the instrument without the shunt is placed in the position G and the contact b is moved along the graduated slide wire until a zero setting is obtained. The objection to this method is that if the millivoltmeter is sufficiently sensitive to be used as a zero instrument it is not likely to be reliable as an ammeter, and if reliable as an ammeter it is not likely to possess sufficient sensitivity when used as a detector or null instrument. Various modifications of this device, however, have proven of value in thermoelectric pyrometry.

Northrup Pyrovolter.—Figure 7 shows the wiring diagram of the Northrup pyrovolter manufactured by the Pyrolectric Instrument Co. Referring to Fig. 7 (a), the dry cell B contained in the case of the instrument sends a current through the variable resistance R and the fixed resistances C and S. The resistance C of copper is equal in value to the resistance of the copper coil of the moving element of the galvanometer. The couple T is connected in series with the moving coil of the galvanometer across the resistance S. The resistance R is adjusted until the galvanometer reads zero by turning the knurled head on the lower right-hand corner of the instrument. The key on the lower left-hand corner is then depressed which throws the thermocouple and the resistance C out of the circuit and replaces C by the galvanometer G of equivalent resistance, Fig. 7 (b). The galvanometer is now deflected by an amount proportional to the current flowing through it, which in turn is proportional to the potential drop

across S. The scale of the instrument is graduated to read the potential drop over S and since this potential difference is made equal to the electromotive force of the couple by the initial setting for zero deflection, the galvanometer indicates directly the true electromotive force of the couple. The initial setting is not altered by introducing resistance into the thermocouple circuit so that the instrument is really a form of potentiometer. The scale may be graduated to indicate temperature for any particular type of couple and the instrument may be obtained with several different scale ranges. The **Northrup continuously deflecting pyrovolter** is the ordinary pyrovolter with the addition of an extra key and an adjustable resistance.

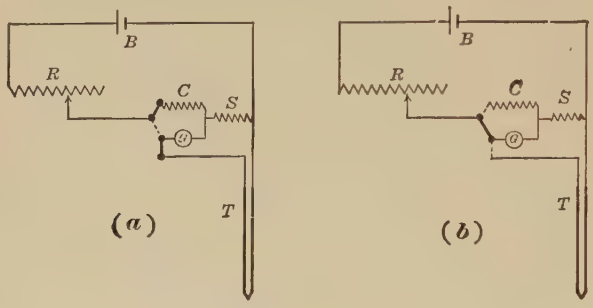


FIG. 7.—Northrup pyrovolter.

Deflection Potentiometer Method.—A potentiometer is ordinarily used as a null instrument. The electromotive force of the couple is exactly balanced by the potential drop over a resistance through which a current from a battery is flowing. No current flows through the couple circuit when the proper setting is obtained. This condition is indicated by zero deflection of the galvanometer in series with the couple. The objection sometimes raised against the ordinary potentiometer for industrial installations is that it requires a manual adjustment of a dial or slide wire every time an observation is made. This objection is practically eliminated in the deflection potentiometer which may be constructed so that it embodies the accuracy of the ordinary potentiometer and the convenience of the galvanometer indicator.

In the deflection potentiometer part of the electromotive force of the couple is balanced against the potential drop over a resistance through which a current is flowing and the remaining portion of the electromotive force is indicated by the deflection of the galvanometer in series with the couple. For example, the instrument may be constructed with a dial of say 16 points representing potential differences from 0 to 15 millivolts and a galvanometer which gives full scale deflection on 1 millivolt. The dial is set to the approximate electromotive force developed by the couple and the dial reading combined with the galvanometer reading gives the true electromotive force of the couple. If the couple developed an electromotive force of 13.58 millivolts, the value 13 is obtained from the dial and the value 0.58 is read from the galvanometer scale. Usually the scale is graduated both ways with zero at the center so that + or − deflections may be readily shown. Thus in the above example, if the dial had been

set at 14 the pointer would have read 0.42, and the final reading would be 14.00 − 0.42 = 13.58 as before. To avoid the necessity of thus subtracting a number from the dial setting, the galvanometer scale may be graduated from left to right as follows:

center

0, 10, − − − 80, 90, 0, 10, − − −100. When the pointer swings to the left the next lower dial reading is used. Hence in the above example with the dial set at 14, the galvanometer would read 58 on the left part of the scale. Thus the correct value is 13.58 as before. In many industrial processes the temperature of the couple will vary only slightly during several hours, so that a new dial setting is infrequently required. Thus the method for obtaining the electromotive force of the couple is nearly as simple as when an ordinary galvanometric indicator is used.

The theory of the deflection potentiometer has been developed in detail by Brooks,[1] and instruments of this type have been designed by Brooks and by White. A compensating resistance is mounted as an integral part of the dial so that turning the dial

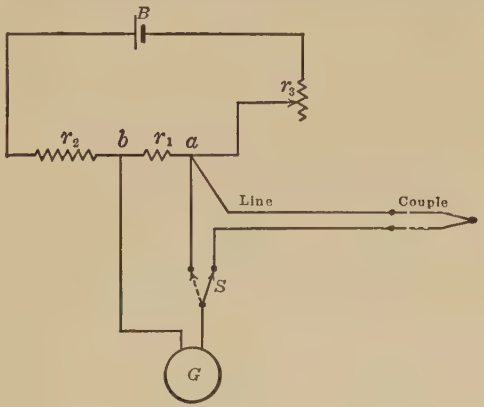


FIG. 8.—Deflection potentiometer.

changes the electromotive force setting and at the same time adjusts the compensating resistance in the galvanometer circuit to its proper value.

Figure 8 illustrates a deflection potentiometer for thermocouples, made by the Taylor Instrument Cos. This instrument is known as the "range control board." The galvanometer G is provided with two scales, in the ranges zero to 500°C. and 450 to 950°C. respectively. The galvanometer circuit is connected at fixed points a and b within the potentiometer, and when the instrument is to operate in the lower range the battery circuit is opened. Thus the potentiometer setting e' is made zero without changing the value of r_1. In this range the instrument operates as an ordinary galvanometric indicator. If the temperature of the thermocouple is about 450 or 500°C., a current of such magnitude is made to flow through r_1 that the potential drop e' across r_1 balances the electromotive force developed by the couple when at 450°C. The temperature will then be indicated on the high range scale.

The total resistance of the galvanometer circuit is almost exactly equal when operating in either range, since when operating in the upper range the shunting effect of $r_2 + r_3$ on r_1 is negligible. Since a separate galvanometer scale is provided for each setting of the potentiometer (o and e') it is not really necessary, however, that the sensitivity be equal in the two cases.

For the high range, the current from the battery is adjusted by connecting switch S as shown by the dotted line and setting r_3 so that the galvanometer deflects to a

[1] Bureau of Standards *Sci. Papers* Nos. 33, 79, 172, 173

marked position. Provision must be made for reversing the direction of the current from the battery through the galvanometer after this adjustment has been made. The figure does not show this or the switch for opening the circuit when the instrument is to be used for the low range.

Figure 9 shows the Leeds & Northrup instrument which is a modification of a design by W. P. White. With proper proportioning of the resistances, the slide wire may be set to read on a graduated scale the exact temperature required. The galvanometer G accordingly indicates the departure of the actual temperature from the temperature desired. It thus serves as a very convenient guide to the operator of a furnace who can see at a glance by how many degrees the temperature at any time differs from the temperature at which the furnace should be operated.

Temperature of the Cold Junctions of Thermocouples.—The electromotive force developed by a thermocouple depends upon the temperature of the cold

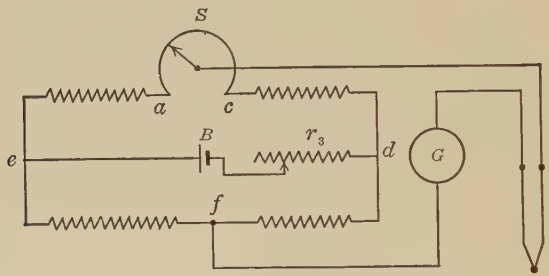


Fig. 9.—Leeds & Northrup deflection potentiometer.

junctions as well as upon the temperature of the hot junction. For certain base metal couples having a nearly linear relation between temperature and electromotive force, the electromotive force is approximately proportional to the difference in temperatures of the hot and cold junctions. With such couples a change of 50°C. in the temperature of the cold junctions, unless corrected for, would result in an error of 50°C. in the final temperature measurement.

If a couple is calibrated with a cold-junction temperature of t_0°C. and is used with a cold-junction temperature of t'_0°C. the true temperature of the hot junction is obtained by adding to the observed temperature the value $(t'_0 - t_0)\ K$ where K is a factor depending upon the particular couple employed and upon the temperature of the hot junction. The factor K varies from 1.5 to 0.3, but for rough work may be assumed 1.0 for base-metal couples and 0.5 for rare-metal couples. The following table gives the cold-junction factors for several different types of couple.

The corrections may be applied directly, without computing, by setting the pointer of the galvanometer to read the cold junction temperature on open circuit. The setting is made by turning the zero-adjustment screw of the indicator when the couple is disconnected. This method of correcting for the cold junction temperature is accurate but requires of course new settings whenever the temperature of the cold junction is altered. Indicators of the potentiometric type frequently have a movable slide on the temperature scale or an auxiliary dial (Cf. Bureau of Standards, *Tech. Paper* No. 170) which when set to the temperature of the cold junction gives perfect compensation at all temperatures of the hot junction. These two compensation methods also require new settings whenever the temperature of the cold junction is altered. With large and permanent installations the applying of corrections for the

TABLE 8.—COLD JUNCTION CORRECTION FACTORS

Engelhard "LeChatelier"		Johnson Matthey "LeChatelier"		Copper-constantan		Iron-constantan	
Temperature, degrees Centigrade	K	Temperature, degrees Centigrade	K	Temperature, degrees Centigrade	K	Temperature, degrees Centigrade	K
265– 450	0.65	250– 400	0.60	0– 50	1.00	0– 100	1.00
450– 650	0.60	400– 550	0.55	50– 80	0.95	100– 600	0.95
650–1,000	0.55	550– 900	0.50	80–110	0.90	600–1,000	0.85
1,000–1,450	0.50	900–1,450	0.45	110–150	0.85	Chromel-alumel	
				150–200	0.80	0– 800	1.00
				200–270	0.75	800–1,100	1.05
				270–350	0.70		

temperature of the cold junction by any of the above methods is frequently troublesome since this temperature may vary considerably in practice over a few hours. There are several methods for obviating the necessity of applying such corrections and of making manual adjustments. The head of the couple may be fitted with a water jacket through which water flows at practically constant temperature from the water main. Copper wires lead from the terminals of the couple inside the water jacket to the indicator, and the pointer of the indicator on open circuit is set once for all to read the mean temperature of the water, usually 10 to 20°C. If the potentiometer indicator is used the zero adjustment slide or the cold junction dial is set at this temperature.

Compensating Leads.—The use of compensating lead wires from the couple to the indicator is in general the most satisfactory method for minimizing the cold-junction errors in industrial installations. For base-metal couples these lead wires are of nearly the same materials as those employed in the couple. For example, with a chromel-alumel couple one lead wire is chromel which is connected to the chromel wire of the couple. The other lead wire of alumel is connected to the alumel wire of the couple. Small stranded wires are used for flexibility. Thus the cold junction is carried away from the head of the couple, where the temperature varies, to a point at some distance from the furnace where the temperature is reasonably constant, and from this point copper wires lead to the indicator. The compensating wires may terminate in a thermostated cold-junction box or may be buried underground. At a depth of 10 ft. beneath the floor of a large building the temperature remains constant to within 2°C. throughout the year. Usually this mean temperature is about 12°C. for temperate climates but may differ from this value somewhat if the location is in the immediate vicinity of a large furnace. To use this method for controlling the temperature of the cold junction, an iron pipe of the proper length, closed at the bottom is driven into the ground and the two cold junctions, for example copper-alumel and copper-chromel, well soldered and carefully insulated are threaded to the bottom of the pipe in such a manner as to be conveniently remov-

able when necessary. The space at the top of the pipe around the couple may be plugged with asbestos or waste and covered with pitch to keep water away from the insulation. The scale of the indicator is set to read the mean temperature of the bottom of the tube. It is convenient to have an extra pair of compensating leads or an extra thermocouple with its junction at the bottom of the pipe to measure this temperature occasionally. Usually the compensating leads of a base metal couple are marked or are equipped with one way terminals, so that they are easily connected properly to the head of the couple. If reversed at the couple the leads will cause an error double the amount of the compensation. When compensating leads of a base metal couple are properly connected to the couple no deflection of the indicator is obtained by heating the head of the couple.

The high cost of platinum prevents the use of compensating leads of the same metal in the case of a rare-metal couple but inexpensive lead wires of copper and an alloy of nickel-copper are now available for use with the platinum-platinum 90, rhodium 10 couple. These lead wires do not compensate individually but taken together they compensate to within 5°C. for a variation of 200°C. in the couple-lead wire junctions. Since the compensating lead wires for the rare metal couple do not compensate individually both terminals on the head of the couple should be always as nearly as possible at the same temperature. The copper wire of the compensating leads is connected to the platinum-rhodium wire of the couple and the copper-nickel wire is connected to the platinum wire of the couple, *i.e.*, alloy wire to pure metal in each case. The cold junction is then located at the indicator end of the compensating leads. The temperature of this junction may be controlled if necessary by one of the methods described above. Copper wires are carried from this point to the indicator.

Many other methods are employed, either with or without compensating leads, for automatically correcting for the cold junction temperature. Leeds & Northrup use several different systems with their potentiometric indicators. Cold junction compensation can be accomplished over a limited range by use of a "shunted couple." The Cleveland Instrument Co. uses a Wheatsone bridge one arm of which contains a nickel coil located in the head of the couple. These methods are discussed by Foote, Harrison and Fairchild in a paper given at the Chicago Symposium of the American Institute of Mining Engineers, September, 1919, on pyrometry.[1]

Thermocouple Installations.—The installation of a large thermocouple equipment requires the services of competent electricians. Just as much attention, if not more, should be given to the wiring, switches, switchboards, etc. as is given in the case of ordinary power installations. Proper fixtures should be used to mount the couple in the furnace. Lead wires should have a weather proof covering and should be run in a metal conduit except for a short length of flexible cable at the ends of the conduit. The conduit should be grounded to prevent leakage from power installations or lighting circuits. All joints in the lead wires should be soldered and taped. When indicators or recorders of low resistance are employed it is of the greatest importance to have a well constructed electrical installation to insure a constant line resistance. Since instruments of low resistance are usually calibrated for a low line resistance of definite value the size of copper wire required for a long line may be as large as No. 12 or No. 10. Frequently switches rated at 100 amp. are required although the actual thermoelectric current is only a few milliamperes. If the indicator is of high resistance or operates upon the potentiometric or semi-potentiometric principle the factor

[1] This has been issued as a separate volume by the Institute.

of very low line resistance is not of great importance, but the wiring should be well installed for the psychological effect if nothing else. Indicating and recording instruments usually should be mounted upon switchboards, except of course in the case of portable indicators, with suitable selective or commutating switches when several couples are used with one indicator.

When the head of the couple is exposed to severe conditions, rain, etc., as in out side kilns, a weather-proof terminal head should be used. This consists of an outside casing which fits over both binding posts. The cover may be tapped for conduit wiring or provided with a packing gland or stuffing box if a length of flexible cable is used between the couple and the conduit. The lead wires should be carried from the couple to the indicator through rooms as cool as conveniently possible. Copper has a high temperature coefficient of resistance and the frequent practice of running wires over the top of a long row of furnaces may cause large variations in line resistance.

The indicator or recorder should be conveniently located. If the instrument is desired especially for the use of the operator of a furnace, it should be placed where it is readily available. It should be mounted where vibration and excessive dirt and dust will not injure the delicate parts of the mechanism. In almost all industrial installations outside protecting cases are required to prevent dust from filtering through the case of the indicator. The various screws and gears of a recorder require occasional attention. The tools for this work, screw-drivers, oil can, etc., may be mounted inside the outer protecting case where they are available for immediate use. Special devices are employed to damp out vibration when this is serious, as in the neighborhood of a trip hammer or rolling mill. Frequently the instruments are suspended on spiral springs. One convenient method especially suitable for heavy instruments such as a recorder, is to mount the instrument on a shelf which is supported on a pier by four ordinary tennis balls, one at each corner of the shelf. The tennis balls damp out vibration very satisfactorily.

The proper location of couples in the furnace depends upon the particular process and use to which the furnace is put. The primary consideration must be to locate the hot junction at the point the temperature of which is required. A secondary consideration, however, is to locate the couple where the lead wires may be conveniently taken care of. The space between the protecting tube of the couple and the furnace wall should be tightly plugged with refractory cement, so that hot air cannot strike through the hole on to the head of the couple, nor cold air be drawn in thus cooling the hot junction of the couple.

The use of extension or compensating lead wires, cold-junction boxes, etc., has been discussed elsewhere. The cold-junction box should be so located as to reduce the amount of compensating lead wire required to a minimum. Compensating lead wires are somewhat costly and should not be employed extravagantly. Also the use of long lengths of compensating wire increases the line resistance since the resistivity of these leads is much higher than that of copper.

In case the cold junction is buried under ground, it must not be located too near a large furnace. Either the distance from the furnace, or the depth at which the junction is buried must be increased. A depth of 10 ft. at at least 10 ft. from a large furnace is usually satisfactory.

The use of a common return wire for a multiple installation is in general unsatisfactory. With such installations, short-circuits through the metal protecting tube of the couple to the furnace and other couples are likely to occur. The trouble which the short-circuits can produce is sufficient to warrant the extra cost of copper required to prevent them. With the common return, leakage from a power installation affects the reading of every couple connected to the return and a leakage through a high resistance may alter the readings of every couple by the same amount so that

the presence of such leaks are not always readily detected. It is also possible by leakage from different couples to the ground to obtain very erratic and erroneous readings when the common return is employed. Base metal couples are frequently constructed with the hot junction welded to the end of the iron protecting tube in order to reduce thermal lag. Even when this welded junction is not made the hot junction usually touches the protecting tube and is in good electrical contact with it especially since at high temperatures, insulation resistance becomes very low. Suppose that the iron tubes of two chromel-alumel couples are grounded to the iron casing of a furnace. The two hot junctions are thus connected to each other by a circuit of iron. The electrical circuit is represented by Fig. 10. On the indicator we have a chromel-alumel couple one leg of which is shunted by an alumel-iron-alumel differential couple. If the temperatures of the hot junctions of the couples are the same, this differential couple produces no effect. It will, however, alter the reading of the indicator whenever the two temperatures differ. Both the chromel-alumel couples will accordingly give erroneous results. If individual returns are used the iron circuit produces no effect. When grounds occur further back from the hot junction, for example between the common return and the other lead wire of a single couple, all couples on the common return have in addition to their own electromotive force an impressed IR drop due to the current flowing in the shunted couple. This may cause a large error in the reading of every couple on the line. Installations employing a common return are extensively used in the industries. It is a dangerous practice and one which should be avoided as far as possible.

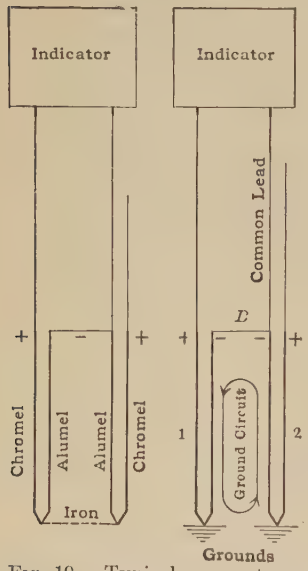


FIG. 10.—Typical pyrometer circuits.

Wiring Diagrams of Thermocouple Installations.—Figure 11 illustrates a simple thermoelectric installation for a rare-metal couple. The couple is properly protected by a porcelain or quartz tube and if necessary by an outer tube of iron, chromel, fireclay, etc. From the head of the couple compensating lead wires are carried to the bottom of a pipe driven 10 ft. under ground. From the bottom of the pipe copper lead wires are carried to the indicator.

Figure 12 illustrates a multiple installation for five thermocouples. In this case a common return is employed although as stated above the use of an individual return is preferred. By use of the common return for this installation four lengths of copper wire from the couples to the recorder have been saved, and the commutating switch is simpler. The indicator for the operator of the furnaces and the recorder for the superintendent's office are mounted in parallel. The indicator or recorder may be connected to any couple desired by setting the commutating switch illustrated in the lower halves of the cases of the instruments. Such an installation can be employed only when the instruments have a high resistance. The recorder and indicator when connected to the same couple at the same time act as a shunt on each other and this tends to make both instruments read low, whereas if the two instruments are calibrated to read correctly in parallel they will both read high when connected to different couples. An example illustrates this point. Suppose the line and couple resistance for each circuit is 3 ohms and the resistances of the recorder and indicator are 500 ohms

each. Assume that both instruments are calibrated to read correctly when connected separately to any couple. The potential drop E across the terminals of either instrument bears the following relation to e the electromotive force of the couple.

$$E = \frac{500}{503} e$$

The scale of the instrument is graduated to take account of this reduction in electromotive force. When the two instruments are connected in parallel, the potential

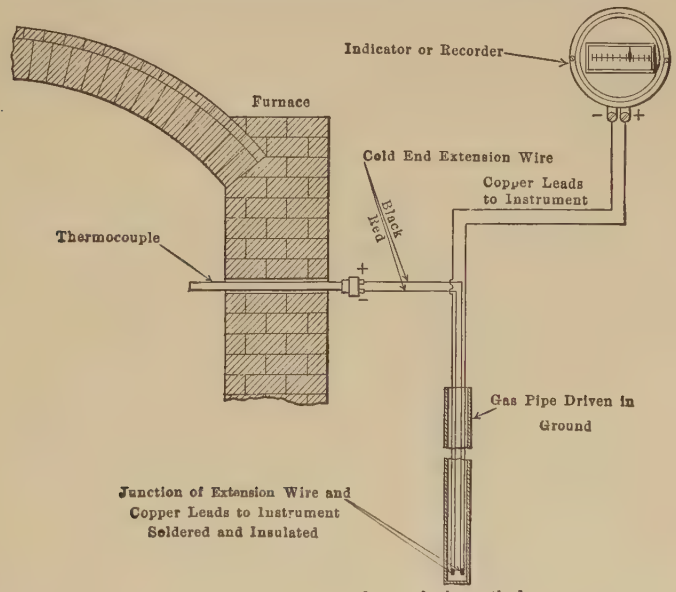


FIG. 11.—Rare-metal couple installed.

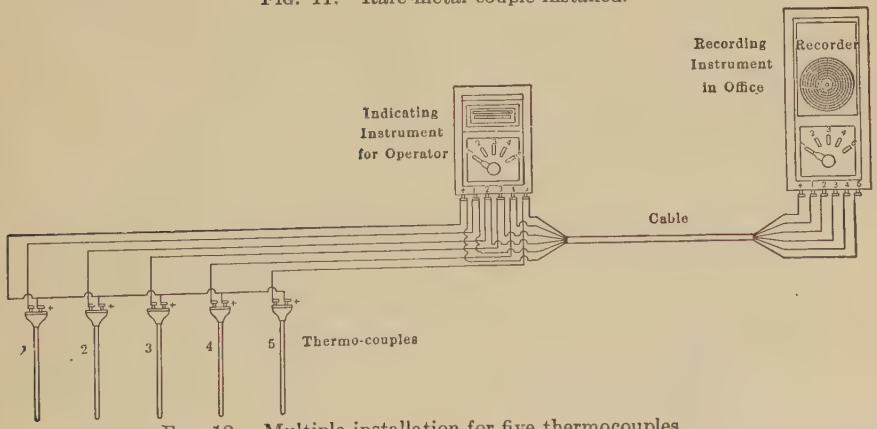


FIG. 12.—Multiple installation for five thermocouples.

drop across the indicator and recorder is given by the following equation where R = resistance of indicator or recorder and r = line resistance.

$$E' = \frac{Re}{R + 2r} = \frac{500}{506} e$$

There is accordingly a reduction in electromotive force by the factor 500/506 when the instruments are connected in parallel to the same couple. Each instrument is

calibrated for a reduction in electromotive force of 500/503. The error resulting from the parallel connection is thus 0.6 per cent or about 6°C. at 1,000°C. Hence, if the operator of the furnace switches couple number 5, for example, on the indicator when this couple is also connected to the recorder, both instruments will read about 6°C. low at 1,000°C. This error is usually insignificant.

A similar example will be considered for an indicator and recorder of low resistance. Let the line resistance $r = 3$ ohms as before and the resistances R of the indicator and of the recorder $= 10$ ohms each. The potential drop E across the terminals of either instrument bears the following relation to e the electromotive force of the couple.

$$E = \frac{Re}{R + r} = \frac{10}{13} e$$

When the two instruments are connected in parallel the potential drop across the indicator and recorder is as follows:

$$E' = \frac{R}{R + 2r} e = \frac{10}{16} e$$

The instruments are calibrated to read correctly when used separately, that is for a reduction in electromotive force by the factor 10/13. When used in parallel there is a reduction of electromotive force by the factor 10/16. The error resulting from a parallel connection is thus 19 per cent or about 190°C. at 1,000°C. Hence if the operator of the furnace switches couple number 5, for example, on the indicator when this couple is connected to the recorder both instruments will read low by 190°C. at 1,000°C. This is a very serious error. Accordingly instruments of low resistance cannot be operated alternately separately and in parallel on the same couple. They must be used *always* either separately or in parallel. In either case the scales of the instruments are graduated for the proper reduction in electromotive force due to line drop. It is frequently the practice to use cut out switches so that when the indicator is set on couple number 5, for example, this couple is automatically thrown out of the recorder circuit. The paralleling of instruments having a resistance of 300 ohms and more, when the line resistance is less than 3 ohms or of potentiometric instruments is a safe practice. The paralleling of instruments of lower resistance requires specially graduated scales or the use of specially wired circuits. Instruments of low resistance designed for parallel operation should not be used separately. Instruments of low resistance operated with cutout switches may be used separately since the switches are designed so that while a recorder and indicator are operated on the same line, they are never connected to the same couple at the same time.

Figure 13 illustrates a multiple thermocouple installation connected to a single indicator. Compensating lead wires are carried from the couples to a conveniently located cold-junction box. The temperature of this box is thermostatically controlled. From the cold-junction box copper wires are carried to the terminal block and selective switch illustrated. A common return has been employed between the cold-junction box and the switchboard. In general it is preferable to use individual return wires for each couple. The switchboard illustrated is designed for six couples. By pressing one of the buttons shown any desired couple is connected directly to the indicator.

For large installations several hundred couples may be connected to a switchboard and the operator of the board connects the couples successively to the indicator. The switchboard is frequently designed somewhat similar to an ordinary telephone distributing board. Often in these large installations communication between the operator of the switchboard and the operator of the furnace is maintained by a system of colored electric lamps. This method of semi-automatic temperature control is meeting with great success in the industries.

Use of Junction Box.—Figure 14 illustrates a wiring diagram for a multiple-couple installation which is very useful in saving compensating lead wire and in

thus reducing the cost and the resistance of the line. The junction box is a cast-iron box such as those used for underground telephone wiring. The box is not thermostated since a constant and measured temperature is not required.

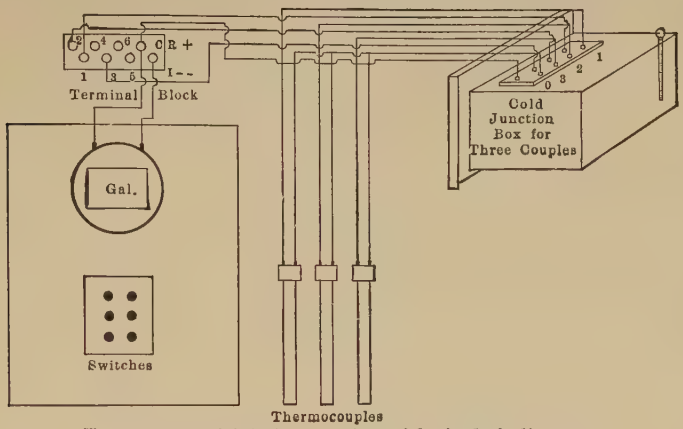


FIG. 13.—Multiple installation with single indicator.

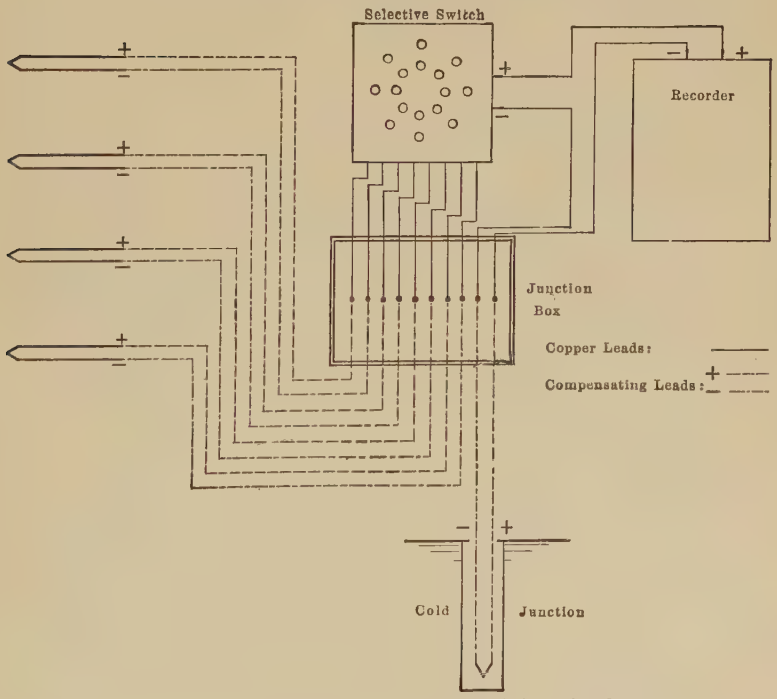


FIG. 14.—Multiple installation with junction box.

It is merely necessary to have a uniform temperature in the box. This is secured when the box is constructed of heavy metal. The electromotive force developed at the junctions of compensating leads and copper leads is compensated for by a common junction in the opposite direction inserted between the selective switch

and the indicator or recorder. A common cold junction is also placed here. In the illustration this is located in a pipe buried 10 ft. underground. The selective switch and recorder or indicator are usually mounted in a single case. The common cold junction and the junction-box compensating couple are connected at the recorder between the switch and the binding-post terminals of the instrument as illustrated. The cold junction is placed near the junction box and the recorder or indicator (with switch if desired) may be any distance away since only copper leads are used from this point to the junction box. The method is especially useful where separate cold junctions require too long compensating leads. The following example illustrates a case where such an installation is desirable. Suppose the temperature of a coke oven, 20 by 40 by 150 ft. is measured by nine couples inserted in the top. The indicator is located at the ground level 40 ft. from the furnace. The buried cold junctions are 20 ft. in front of the oven. The following illustrates the amount of compensated lead wire required to reach the buried cold junctions when the junction box is not employed.

Three couples at rear of furnace	3(150 + 20 + 20 + 10) ft.
Three couples at center of furnace	3(75 + 20 + 20 + 10) ft.
Three couples at front of furnace	3(20 + 20 + 10) ft.
Total compensating cable	1,125 ft.

Consider the same installation when a junction box is located on top of the furnace at the center.

Three couples at rear of furnace	3(75) ft.
Three couples at center of furnace	2(10) ft.
Three couples at front of furnace	3(75) ft.
From box to cold junction	(75 + 20 + 20 + 10) ft.
Total compensating cable	595 ft.

By means of the junction box we effect a saving of some 500 ft. in the compensating cable and need to bury only one pair of junctions, and just as satisfactory an installation is obtained. In installing a large multiple-couple equipment with a junction box, it is very important to insure that the common cold-junction couple is connected with the correct polarity as illustrated. Although we have used a common cold junction for all couples we have not employed the objectional common return. Individual returns are used with every couple shown in the diagram.

In case the recorder is placed where the temperature is quite uniform from day to day, the use of a buried cold junction or thermostated cold-junction box is not absolutely essential. The electromotive force generated at the junction box in Fig. 14 is then compensated for by running one pair of compensating leads from the recorder to the junction box, taking care to connect the negative lead to the negative terminal of the recorder and the positive lead to the selective switch. A simple installation of this kind is illustrated by Fig. 15. Here only one couple is shown, but as many couples as desired may be connected to the multiple-pole selective switch. The compensating lead wires are soldered together inside the junction box. The auxiliary couple formed by the compensating leads is in series with the couple connected in by the selective switch. The cold junction is accordingly at the recorder where the temperature is fairly constant. Changes in temperature of the distributing or junction box thus will not affect the reading of any couple.

Determination of Temperature of Buried Cold Junction.—Several methods are available for obtaining the temperature at the bottom of the junction well. The simplest is to use a thermocouple consisting of the compensating

leads. Insert this in the well, connect it to a portable indicator and measure the secondary cold-junction temperature at the indicator with a thermometer. With a well 10 ft. or more deep and properly located, it is necessary to measure the temperature possibly only once a month. Another method is to lower a thermometer into the well. The thermometer is wrapped with a few layers of cloth leaving the stem exposed near the expected reading. It is left in the well for 30 min. and read quickly after raising. The recorder or indicator is adjusted

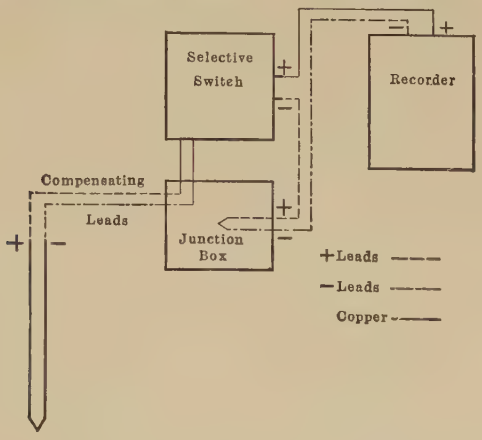


FIG. 15.—Simple junction box installation.

to correct for this temperature or the records may be marked "cold junction" and corrections applied later.

All buried leads to the cold junction should be water-proof insulated, and the junction well should be made watertight. The compensating leads, particularly those for base-metal couples, will generate a large voltaic electromotive force if they are wet and are not insulated with water-proof covering.

Depth of Immersion of Couples.—Thermocouples immersed in furnaces of the various industrial types must be carefully protected. Heavy iron tubes and frequently larger auxiliary protecting tubes of various materials are employed. The thermocouple calibrated in the laboratory will give the temperature of the hot junction when installed in the protecting tubes but there is no certainty that this temperature is that of the furnace. Conduction along the protecting tubes may be so great that the temperature of the couple is far below that of the furnace. Conduction losses affecting the temperature of the hot junction may be reduced and even eliminated by using a sufficient depth of immersion but it is not always possible to do this and it is difficult to determine when the depth of immersion is sufficient. For example, an immersion of 5 cm. might be satisfactory for one type of furnace or for a molten bath, and 30 cm. insufficient for a certain muffle furnace. Each particular case requires special consideration.

The general practice is to use as deep immersion as is conveniently possible and trust that this is satisfactory. Two methods may be suggested for investigating the question, but neither one is conclusive. First, remove the couple alone from the fixed installation, leaving all protecting tubes in place. If it is impossible to remove the iron tube from the couple use a similar tube in the fixed installation without the couple. Explore the temperature inside the protecting tube with an unprotected

couple. If the temperature for several centimeters along the axis of the tube at the inner end is practically uniform, the depth of immersion is satisfactory. If, however, the temperature falls rapidly in this region the immersion is not deep enough. Second, the couple previously standardized is mounted complete in the fixed installation and compared with another checking couple mounted along the side of the permanently installed couple. The checking couple must have a small cross-section and must be used either unprotected or protected by an extremely thin tube in order to minimize the loss of heat by conduction. The hot junctions of this couple and of the couple under test are brought closely together but not in contact. The checking couple should indicate a temperature more nearly that of the furnace than the temperature observed with the fixed couple. If the temperature difference between the two couples is large a greater depth of immersion should be employed. These methods are complicated by local variations in the temperature of the furnace but checks of this nature, although somewhat unsatisfactory, are better than no checks at all.

Couples Purposely Insufficiently Immersed.—It is frequently desirable purposely to immerse the couple to an insufficient depth. In many processes the furnace is operated at such a high temperature that a thermocouple or protecting tube cannot withstand the severe conditions to which it may be subjected. In this case the couple may be immersed only part way through the furnace wall, or to a distance flush with the inner wall of the furnace. The temperatures indicated by couples installed in this manner are always lower than those of the furnace interior, but they bear a fairly definite relation to the temperature of the furnace, and hence the method is satisfactory for temperature control and reproduction of furnace conditions from day to day.

Protecting Tubes for Thermocouples.—The choice of a proper protection tube for a thermocouple is nearly as important as the selection of the material for the couple. Among others the following properties of a protection tube should be considered in reference to the particular process for which the tube is required:

(*a*) Low porosity to gases: Many tubes become very porous to furnace gases at high temperatures. Furnace gases usually attack the couple.

(*b*) Low volatility: Certain metal tubes are undesirable at high temperatures because the metal distils upon the couple, thus altering its calibration.

(*c*) Ability to withstand high temperatures.

(*d*) Ability to withstand sudden changes in temperature.

(*e*) Ability to withstand mechanical shocks and strains.

(*f*) High rigidity or viscosity: Protecting tubes frequently deform and exhibit the phenomenon of plastic flow at high temperatures.

(*g*) Thermal conductivity: High thermal conductivity is frequently desirable when rapidly changing temperatures are measured. Usually, however, low thermal conductivity is desired so that the flow of heat along the tube is as small as possible.

(*h*) Ability to resist corrosion from molten metals or furnace gases.

Excellent protecting tubes are obtainable for many different industrial processes. However, for certain processes, satisfactory tubes have not as yet been developed. The problem is one for the ceramist or metallurgical engineer. This is particularly true in obtaining suitable tubes for molten metals, especially iron and brass. The following describes the more important protecting tubes now in use industrially.

Fused quartz or silica tubes are used as insulators through which the wires of a rare-metal couple are threaded, and as outer protecting tubes. The molten quartz may be drawn and worked like glass or the material may be ground and pressed, with a suitable binder, into the proper form and fired in a manner similar to that employed for the manufacture of porcelain. These tubes are translucent like china. Trans-

parent quartz-glass tubing is manufactured, but on account of its cost is not used industrially for thermocouple protection. Fused quartz affords good protection up to 1,050°C. in an oxidizing atmosphere free from alkalies. The material is somewhat pervious to hydrogen and probably to other reducing gases, but is not pervious at this temperature to oxygen or carbon dioxide. Any reducing gas within the protecting tube of a rare-metal couple is disastrous, particularly when the tube contains silica. The silica is reduced to silicon which is readily absorbed by platinum. Above 1,050°C., and even at lower temperatures after prolonged heating, quartz devitrifies and crumbles away. Quartz tubes withstand sudden changes of temperature without breaking. Heavy sintered quartz tubes, with walls 1 or 2 cm. thick, are sometimes used for extra protection, for example against acid fumes.

Porcelain is used primarily for protection of rare-metal couples. Previous to 1914 a highly refractory porcelain known as Marquardt was imported from Germany. A better grade of this material was developed through the research work of the Bureau of Standards and is now manufactured by Stupakoff under the trade name "Usalite," and by Engelhard under the trade name "Impervite." These two porcelains have a melting point above that of platinum. However, they are impervious to gases only when glazed. The softening point of the glaze used is about 1,300°C. If the tubes are glazed on the outside only they are serviceable as pyrometer protection tubes up to 1,500°C. The glaze on the German tubes softens at 1,200°C. The insulating tubes are not glazed. Pyrometer porcelain for use at lower temperatures is made by Guernsey Earthenware Co., Herold China & Pottery Co., and other American companies, the advertisements of which appear in the trade journals, also by the Royal Worcester Co., England, and a porcelain known as S. C. P. is made by a Japanese firm. The glaze on the Japanese porcelain softens at about 1,100°C. and its maximum serviceable temperature is about 1,250°C., which is higher than that of the Berlin porcelain of similar grade, or the Royal Worcester. We do not have data on similar grades of American porcelain which have been developed during 1918–1919. An unprotected porcelain tube suddenly thrust into a furnace at 1,000°C. will usually break. If it is inserted slowly, however, there is little danger of breakage.

In permanent installations quartz and porcelain tubes and also the iron or chromel tubes of base-metal couples are frequently further protected by heavy outer tubes of fire clay, carborundum, graphite, etc. The outer tube is usually cemented in place in the furnace wall forming a well into which the couple is inserted. In case the outer tube introduces too large a temperature lag or where there is danger of lowering the temperature of the hot junction of the couple by conduction of heat through the heavy tube, the latter is made open at both ends. The couple and its smaller protecting tube is so mounted that the hot junction projects beyond the end of the outer protecting tube a few centimeters.

Carborundum, or silicon carbide, SiC, is used for outer protecting tubes. It has a high-thermal conductivity (about twice that of silica), a low coefficient of expansion (about one-half that of fused alumina), and a high mechanical strength. When heated in an oxidizing atmosphere, oxidation begins at about 1,200°C. Gases except chlorine do not act on carborundum. Carborundum reacts with practically all metals at high temperature. Platinum must be thoroughly protected from the carborundum by a gastight inner tube. Carborundum or silfrax, which is pure carborundum of very fine crystalline structure, is highly satisfactory for ordinary furnace work. It is sometimes used in molten glass and open-hearth slag, although basic slags attack it readily.

Cast nichrome or chromel tubes are used extensively for protection of both base-metal and rare-metal couples. These tubes resist oxidation remarkably well and although much more costly than iron tubes, in many processes, their longer life war-

rants and requires their use. Chromel "A" may be used continuously up to 1,200°C. The tubes are heavy enough to withstand considerable mechanical strain. It has not been possible as yet to draw these alloys into tubes. The material cannot be readily machined, but when necessary can be finished off on an emery wheel. It is also possible to cut threads for pipe fittings. In order to economize on the use of these materials, iron tubes are frequently employed having short tubes of nichrome or chromel welded to them. Only the part of the protecting tube exposed to the furnace need be of the expensive alloy.

In processes carried out at low temperatures where either iron tubes or chromel and nichrome tubes may be employed, experiments should be performed to determine the relative value of these tubes in life in hours per dollar of cost. There are practically no data available in this regard. In heat-treating furnaces and carbonizing furnaces, chromel "A" is often used. Chromel "C" and nichrome last many months in lead baths. Chromel and nichrome do not volatilize so readily as iron. Base-metal couples are thus better protected by these tubes than when iron or steel is employed. Chromel "A" contains practically no iron. Chromel "C" and nichrome have a rather high iron content.

Graphite tubes afford an excellent protection to quartz or porcelain tubes on rare-metal couples, and are frequently used with base-metal couples for molten metals. Porcelain encased in a sheath of graphite can be used in molten aluminum. Platinum couples must be thoroughly protected against the vapors distilled from graphite or carbon and from the reducing atmosphere present near heated graphite.

Fireclay outer protecting tubes are used for protection in kilns, glass and steel furnaces, annealing ovens, etc. Usually they are mounted vertically in the top of the furnace and may be cemented in place. Small fireclay insulating tubes are used on base-metal couples.

Corundite consists of emery with a plastic clay binder. It is used in ceramic and glass industries for outer protection tubes.

Natural corundum usually contains a large amount of iron. The artificial product, fused Al_2O_3, known under the trade name of alundum, is practically free from iron. This material is very desirable for protecting rare-metal couples and for use also as outer protecting tubes. The tubes are made from ground Al_2O_3 mixed with a clay binder. The more refractory the tubes, the less impervious to gases they become. The inner protecting tubes are glazed in order to reduce the porosity and the glaze is coated with an outer layer of alundum. This method of preparation permits the use of the tubes at temperatures above the softening point of the glaze. Such tubes are serviceable up to 1,400°C. Outer protecting tubes without glazing are made to withstand temperatures up to 1,550°C. and even higher. Alundum is mechanically strong and resists temperature changes much better than porcelain.

Pure nickel is frequently used for cyanide baths. In an oxidizing atmosphere a thick tough coating of the oxide forms which does not readily scale. The oxide thus affords protection against further corrosion.

Seamless-steel and wrought-iron tubes are usually furnished with base-metal couples. They are satisfactory for many processes up to 800 or 900°C., for example in a muffle furnace.

Calorizing is a process by which the surface of a wrought-iron tube is impregnated with metallic aluminum. Calorized tubes resist oxidation better than the pure iron or steel tubes.

Duriron, a high-silicon iron alloy, is sometimes used at lower temperatures as a protection against acid fumes. When subjected to sudden temperature changes the material may fracture and some observers report it as extremely brittle.

TABLE 9.—CALIBRATION DATA OF REPRESENTATIVE COUPLES

Cold-junction temperature = 0°C. electromotive force in millivolts

Engelhard "Le Chatelier"		Johnson-Matthey "Le Chatelier"		Copper-Constantan		Iron-Constantan			Chromel-Alumel	
Electromotive force	Temperature, degrees Centigrade	Electromotive force	Temperature, degrees Centigrade	Electromotive force	Temperature, degrees Centigrade	Electromotive force	Temperature, degrees Centigrade		Electromotive force	Temperature, degrees Centigrade
0	0	0	0	0	0	0	0	0	0	0
1	147	1	146	1	25	5	105	95	5	122
2	2 5	2	260	2	49	10	204	186	10	243
3	374	3	364	3	72	15	299	277	15	363
4	478	4	461	4	94	20	392	367	20	482
5	578	5	553	5	115	25	483	457	25	601
6	675	5	641	6	136	30	574	546	30	721
7	770	7	725	7	156	35	662	632	35	844
8	861	8	806	8	175	40	749	713	40	970
9	950	9	884	9	194	45	836	792	45	1,100
10	1,037	10	959	10	213	50	924	871		
11	1,122	11	1,032	11	232	55	1,011	950		
12	1,206	12	1,103	12	250	60	.....	1,030		
13	1,290	13	1,173	13	268	..	*B*	*L*		
14	1,373	14	1,242	14	285					
15	1,455	15	1,311	15	302					
..	.....	16	1,379	16	319					
..	.....	17	1,447	17	336					
..	.....	..	.....	18	353					

B represents mean calibration by Bureau of Standards of Iron-constantan couples from all sources. *L* represents mean calibration of Leeds and Northrup's Iron-constantan couple.

General Theory of Optical and Radiation Pyrometry.—The temperature of a material may be obtained from a measurement of the intensity of the radiant energy it emits. This measurement may refer to the radiation of all wave lengths emitted by the material, or if the material is glowing, the measurement may refer to the visible light emitted, or to the radiation in a very restricted portion of the visible spectrum. However, in general, the intensity of radiation depends not alone upon the temperature of the source, but also upon the particular material constituting the source. Thus glowing carbon appears to the eye about three times as bright as glowing platinum when both are at the same temperature. This is technically expressed by the statement that the emissive power or emissivity of carbon is about three times that of platinum. A material having the highest theoretically possible emissivity is known as a "black body." In general it is customary to assign a numerical value of 1 to the emissivity of a black body. Hence all other materials have an emissivity less than 1. A black body is experimentally realized by uniformly heating a hollow enclosure and observing the radiation coming from a small opening in the wall. The intensity of radiation emitted from this opening depends only on the temperature of the walls. It does not depend upon the material of which the walls are constructed. If E = the emissivity of any non-transparent material and R = its reflection coefficient it can be shown that $E + R = 1$. If a material having an emissivity of say 0.40, and hence a reflection coefficient of 0.60, is placed inside a black body it becomes indistinguishable from its surroundings. The total intensity of radiation leaving the material is the same as that emitted by the black body. Thus while the

material actually emits only 40 per cent of the intensity of a black body at the same temperature, under the above conditions 60 per cent of the radiation falling upon it from the walls of the enclosure is reflected, with the net result that the object appears of the same intensity as its surroundings.

However, if the material is removed from the black body and placed in the open air, the reflected intensity is no longer present and the object appears but 40 per cent as bright as a black body at the same temperature. Optical and radiation pyrometers are usually calibrated to read correctly when sighted upon a black body. Fortunately many technical processes are carried out under black body conditions. Muffle furnaces, many annealing furnaces, etc., are sufficient approximations to "black bodies" to give practically correct temperature readings with the optical or radiation pyrometer. Some materials in the open are nearly "black," for example the oxide formed on iron and steel ingots, rails, etc.

In general, however, corrections must be applied to the pyrometer readings to obtain the correct temperatures of materials in the open. These corrections are very large in the case of clean molten metals. The presence of an oxide film on the molten metal surface greatly reduces the corrections. For temperature control it is not always necessary to correct the observed readings. As far as the factor of emissivity is concerned the actual pyrometer readings although known to be too low, will always be too low by the same amount from time to time for the same observed temperature, and hence will furnish as good information for temperature control and uniformity as could the true temperatures.

Optical Pyrometer Temperature Scale.—The temperature scale for the optical pyrometer is based upon Wien's law for the distribution, in the spectrum, of the energy of a black body. This law may be stated by the following equation where λ denotes the wave length in microns, c_2 a constant = 14,350, ϑ the absolute temperature of the black body, J_λ the intensity at the wave length λ (*i.e.* at a particular color such as red), and c_1 a constant, the value of which is of no moment in pyrometry since, as will be seen, it disappears from the actual working equation

$$J_\lambda = c_1 \lambda^{-5} e^{\frac{-c_2}{\lambda \vartheta}} \tag{1}$$

for a black body. The intensity of radiation $J_\lambda{}^1$, of wave length λ, from a non-black body of temperature ϑ and emissivity E_λ is given by equation

$$J_\lambda{}^1 = c_1 E_\lambda \lambda^{-5} e^{\frac{-c_2}{\lambda \vartheta}} = c_1 \lambda^{-5} e^{\frac{-c_2}{\lambda S_\lambda}} \tag{2}$$

for a non-black body. In the third term of (2) we define S_λ as the apparent temperature in degrees absolute of the non-black body. This is the temperature measured by the optical pyrometer and is less than the true temperature ϑ for all materials except black bodies, when it becomes equivalent to ϑ. From (2) we have:

$$\frac{1}{\vartheta} - \frac{1}{S_\lambda} = \frac{\lambda \log E_\lambda}{0.4343 c_2} = \frac{\lambda \log E_\lambda}{6232} \tag{3}$$

Thus knowing λ and E_λ, it is always possible to obtain the true temperature ϑ from the observed temperature S_λ.

An optical pyrometer is simply a photometer using monochromatic light (usually red) in which the intensity of radiation from either a standard or a constant source (electric lamp, oil flame, etc.) is compared with that from the object of which the temperature is desired. Frequently the two intensities are made equal by adjusting various types of absorbing devices (absorption glasses, iris diaphragm, etc.) interposed either on the furnace side or the standard-lamp side of the pyrometer, depending upon

which source is normally the brighter. In this process of comparison the term $c_1\lambda^{-5}$ of equation (1) is embodied as one of the calibration constants of the instrument.

Radiation Pyrometer Temperature Scale.—The temperature scale for the radiation pyrometer is based upon the Stefan-Boltzmann law expressing the relation between the total energy J radiated per unit time per unit area by a black body and its absolute temperature, $\vartheta°$ absolute, as follows:

$$J = \sigma(\vartheta^4 - \vartheta_0^4) \tag{4}$$

Where ϑ_0 denotes the absolute temperature of the surroundings or of the measuring instrument receiving the radiation, and σ an empirical constant. In general ϑ_0^4 is negligible in comparison with ϑ^4 so the above relation becomes:

$$J = \sigma\vartheta^4 \tag{5}$$

For a non-black body we have:

$$J' = \sigma E\vartheta^4 = \sigma S^4 \tag{6}$$

where E is the total emissivity and S is the apparent absolute temperature of the object sighted upon as measured by the radiation pyrometer. From (6) one obtains:

$$E = S^4/\vartheta^4 \text{ or } \log E = 4\ (\log S - \log \vartheta) \tag{7}$$

Thus knowing the total emissivity E of any material it is possible to obtain the true temperature ϑ from the apparent temperature S as measured by a radiation pyrometer.

Summary Statement of the Two Radiation Laws.—Equation (1) states that the intensity of radiation of a fixed wave length from a black body is proportional to $e^{\frac{-\text{constant}}{\vartheta}}$.

Equation (5) states that the total radiation of all wave lengths emitted by a black body is proportional to ϑ^4. These two laws which form the basis of optical and radiation pyrometry respectively are in agreement with the temperature scale defined by the gas thermometer up to 1,550°C., the upper limit at which a gas thermometer has been used satisfactorily. Above this range to 2,500°C. the scales defined by these two laws have been found, experimentally, to be in mutual agreement, and it is believed that they correctly represent the thermodynamic scale for all temperatures.

Fery Optical Pyrometer.—Figure 16 illustrates the principle of the Fery optical pyrometer. G is a means for producing a divided photometric field. In the later instruments a Lummer-Brodhun or silver strip cube is employed. Part of the field of view is illuminated by the source sighted upon and part by the gasoline lamp L which burns at a constant brightness. By moving the wedges of black glass, pp', the thickness of absorbing glass in the line of sight can be varied until the part of the field illuminated by the source has the same brightness as that illuminated by the lamp. A red-glass screen is used in the ocular so that fairly monochromatic light of this color (0.65μ to 0.63μ) is compared. The relation between the thickness of the wedges x read on a scale and the absolute temperature ϑ is $x + P = \frac{Q}{\vartheta}$ where P and Q are constants determinable by two calibration points. The instrument must be focused upon the radiating source but no corrections for sighting distance need be applied. The Le Chatelier optical pyrometer is similar in principle but is not of constant aperture and important corrections must be made with change of focus.

The Shore pyroscope, Fig. 17, operates upon a principle very similar to that of the Fery optical pyrometer. The instrument has a scale graduated to read temperatures directly, which is a material advantage. The design of optical parts is rather un-

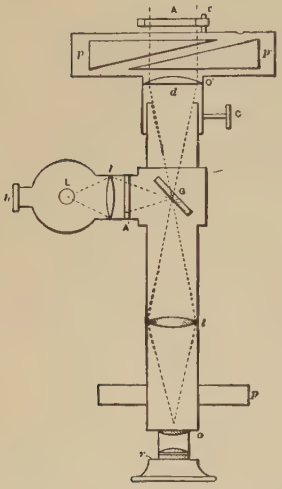


FIG. 16.—Fery optical pyrometer.

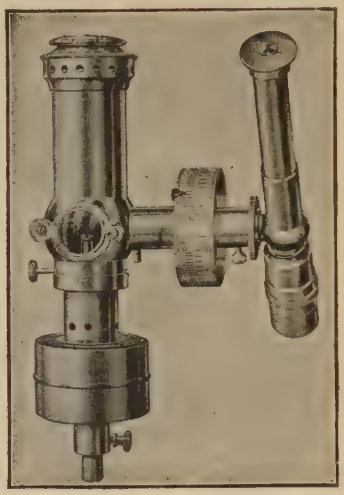


FIG. 17.—Shore pyroscope.

necessarily complicated and it is difficult to match the two fields on account of color differences.

Wanner Pyrometer.—Figure 18 illustrates the arrangement of the optical parts in the Wanner pyrometer. The comparison light is a six-volt incandescent lamp illuminating a glass matt surface in front of the slit S_2. The slit S_1 is illuminated by the source sighted upon. Light from each slit passes through the

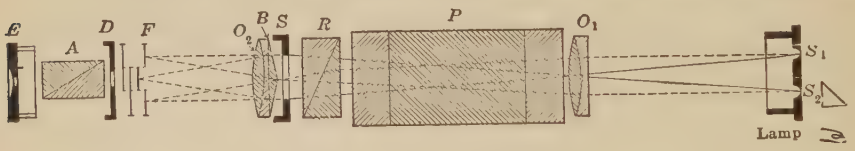


FIG. 18.—Wanner pyrometer.

collimating lens O_1, the direct vision spectroscope P, a Wollaston prism R, a bi-prism B, the second collimating lens O_2, and is brought to a focus at F. The Wollaston prism produces two images of each slit which are polarized at right angles to each other. The bi-prism again doubles the number of images so so that there are finally four images of each slit at F. Six of these images are diaphragmed off by the screen D. The two remaining images, one of each slit, are superposed and are polarized at right angles to each other. From this point the light passes through the nicol prism A and the ocular lens E. The direct vision spectroscope is so adjusted that only red light of wave length about $\lambda = 0.65\mu$ reaches the eye, the other colors being diaphragmed off by the screen D. The ocular is focused on the dividing edge of the bi-prism B. The eye perceives a circular photometric field half of which is illuminated by the slit S_1 and half by the slit S_2. The light from the two fields is plane polarized and the plane of polarization in one field is at right angles to the plane of polarization in the other field. Consequently on rotating the nicol

prism A, one field increases and the other field decreases in intensity. A setting is obtained when the two fields match.

In order to determine the proper brightness at which to operate the electric lamp illuminating the slit S_2, the pyrometer is sighted on a source of standard brightness. This consists of an amyl-acetate lamp with a flame gage having a window of ground glass. The flame is adjusted to a specified height. The ground glass window illuminates the slit S_1. The analyzer nicol A is set at a specified normal point or angle marked on the instrument. The current through the electric lamp is then varied by means of a rheostat until the two fields are matched, and the current is read from the ammeter. This process should be repeated several times and a mean value of the current settings obtained. In using this instrument the current is adjusted to this mean value. The electric lamp burns at a high temperature and consequently deteriorates noticeably. Hence the above adjustment on the normal point requires frequent redetermination. For high precision the adjustment should be made both before and after a series of temperature readings. In the industrial plant once a day or once a week is sufficient depending upon the amount of use.

The calibration of the instrument follows the law

$$\log \tan \varphi = a + b/\vartheta$$

where φ is the angular reading of the analyzer, ϑ the absolute temperature, and a and b empirical constants. The relation between log tan φ and $1/\vartheta$ is linear. Two calibration points serve to determine a and b and a table or plot may be made of φ *vs* t°C. ($t = \vartheta - 273$). Usually such a table is furnished with the pyrometer, or the instrument may be sent to the Bureau of Standards for calibration.

The instrument described above is satisfactory for temperatures greater than 900°C. In the temperature range 700 to 900°C. the intensity of light from the furnace sighted upon is insufficient to permit accurate settings. Hence for temperatures from 700° up the direct-vision spectroscope P is replaced by a red-glass screen, or the objective lens O_1 is made of red glass, and the slits S_1 and S_2 are of much wider opening.

On account of stray light the Wanner pyrometer is not accurate at very small or very large angular readings. Moreover the temperature increases so fast at large angles that the angles would have to be observed with extreme precision in order to obtain any accuracy when expressed in degrees of temperature. The range of the instrument is thus confined to from about 10 to 80 angular degrees. The following table illustrates the relation between angular readings and temperature for a particular instrument. Different instruments may have entirely different or practically identical calibrations as desired depending upon the adjustment of the normal point.

TABLE 10

Angle, degrees	Temperature, degrees Centigrade	
	Without absorption glass	With absorption glass
10	900	1,390
20	990	1,575
30	1,060	1,725
40	1,120	1,865
50	1,185	2,020
60	1,255	2,200
70	1,360	2,460
80	1,535	3,030

Thus the above instrument is satisfactory for temperatures up to 1,500°C. For higher temperatures an absorption glass is mounted in front of the slit S_1 which decreases the light from the furnace in a known ratio. The third column shows the

temperatures which may be measured when an absorption glass of suitable density is employed. If still higher temperatures are desired a denser absorption glass must be used.

With the Wanner pyrometer the tip of the flame of the amyl-acetate lamp should burn level with the top of the flame gage. The setting on the normal point is tedious work as the flame flickers over the field. A screen of black paper placed around the lamp helps to reduce the flicker. The observations must be made in a closed room free from drafts. Any error in the setting of the normal current is carried over to the final temperature measurements so that it is exceedingly important to exercise all possible care in these preliminary adjustments. Examine the screen of the flame gage to assure that no smoke has deposited upon it. A slight film of smoke from the lamp may cause an error of 100° or more. The amyl acetate used in the lamp need not be of high purity. Any good grade of amyl acetate is satisfactory.

From experience with several hundred instruments in use in the technical industries it is evident that these pyrometers are subjected to great abuse. The instrument is made up of delicate optical parts and should not be allowed to become heated. Many of the parts are set in wax and the various optical surfaces are cemented by Canada balsam. The Wollaston prism and the nicol prism in the rotating eye piece are made of calcite. This material is very soft and, although it has the appearance of glass, may be scratched with the finger nail. In practically half the instruments examined these parts have been deeply cut by knives or pointed steel tools. All persons using this pyrometer should be cautioned not to touch any optical part except the lens in the eye cup which requires occasional cleaning. Do not change the setting

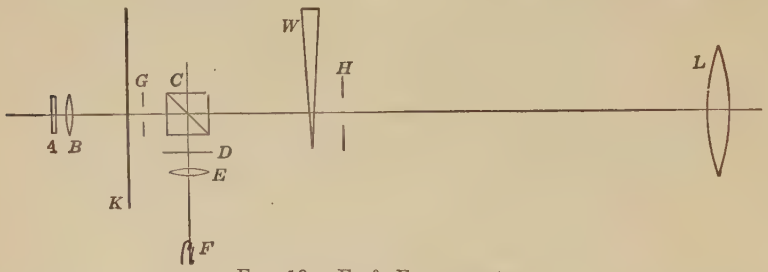


FIG. 19.—F. & F. pyrometer.

of any screw. The moving of a single screw may throw out the adjustment of the pyrometer and cause errors of 500°. If the position of any screw on the body of the instrument is altered, do not attempt to readjust the instrument but return it to the maker. Also do not take the instrument apart to find out what is wrong unless you are perfectly familiar with the optics of such a polarizing system, for it will gain nothing and the chances are that damage will be done. The replacement of the electric lamp will not alter the calibration of the pyrometer.

The Scimatco pyrometer, formerly sold by the Scientific Materials Co., is an improved form of the Wanner pyrometer. All but one of the screws, the tampering with which affects the calibration of the instrument, are enclosed in a metal sheath, and hence are not readily accessible. The instrument has both an angular scale and a scale graduated directly in degrees of temperature. With the Wanner or Scimatco pyrometer the observer can not see through the instrument the object sighted upon. This may cause inconvenience if it is desired to measure the temperature of a small crucible in a furnace.

Figure 19 illustrates the arrangement of optical parts in the F. and F. pyrometer made by the Scientific Materials Co. Light from the furnace is focused at the center of the silver strip cube *C*. This cube produces a circular field divided

through the middle. One-half of the field receives light from the furnace and the other half from the ground-glass screen D which is illuminated by the electric lamp F through the condenser lens E. The ocular containing the red-glass screen A and lens B is focused on the dividing edge of this photometric field. G and H are diaphragms which limit the cone of rays employed. The two fields are matched by turning a thumb screw which moves the black-glass wedge W across the path of the light from the furnace. By a system of gears this movement is transferred to a circular scale on the dial K of the instrument. In appearance the pyrometer resembles the Scimatco, and is used in the same manner. For a normal point setting the pointer is adjusted to read the normal angle and after removing the tube carrying the lens L the instrument is clamped in its case. The flame gage of the amyl-acetate lamp is so mounted that its ground-glass window is adjacent to the diaphragm H. A table is furnished with the instrument giving the relation between the scale reading in angular degrees and degrees of temperature. This instrument is so designed that the object sighted upon is clearly imaged, a distinct advantage over the Wanner pyrometer. The relation between the scale reading α and the absolute temperature ϑ is $\alpha + P = Q/\vartheta$ where P and Q are constants determinable by two calibration points. In a recent modification of this instrument, the lamp is mounted in the line of sight, replacing C, and a semi-circular wedge is employed, so that $\cos \alpha + P = Q/\vartheta$

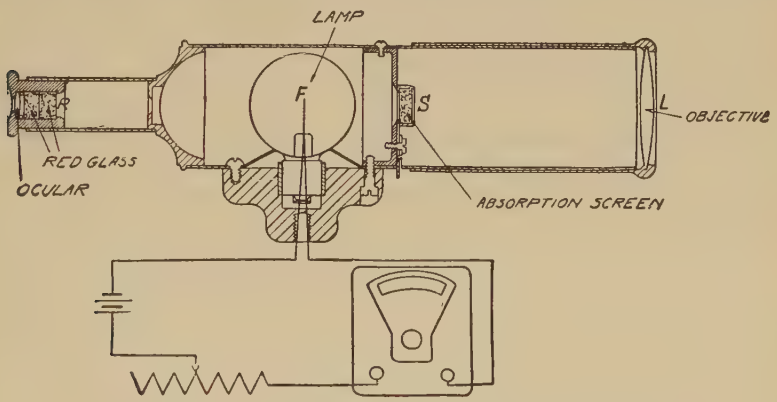


FIG. 20.—Disappearing filament optical pyrometer.

Morse, Holborn-Kurlbaum, and Leeds & Northrup Optical Pyrometers—the filament of a small electric lamp F, Fig. 20, is placed at the focal point of an objective L and ocular forming an ordinary telescope which superposes upon the lamp the image of the source viewed. Red glass such as Corning "high-transmission red" is mounted at the ocular to produce approximately monochromatic light. In making a setting the current through the lamp is adjusted by means of a rheostat until the tip or some definite part of the filament is of the same brightness as the source viewed. The outline or detail of this section of the filament is then indistinguishable from the surrounding field. The current is read on an ammeter and the corresponding temperature is obtained from a plot or table of current versus temperature. The relation between the current, i, through the lamp and the temperature, t°C., is of the form:

$$i = a + bt + ct^2$$

where a, b, c are constants requiring for their determination at least three standardization points.

The lamps should not be operated at temperatures higher than 1,500°C. on account of deterioration of the tungsten filament. If this temperature is not exceeded the calibration of the lamp is good for hundreds of hours of ordinary use. For higher

temperatures absorption glasses, S, Fig. 20, are used between the lamp and objective, or in front of the objective, to diminish the observed intensity of the source. The relation between the temperature of the source $\vartheta°$ absolute and the observed temperature $\vartheta_0°$ absolute measured with the absorption glass interposed is as follows:

$$\frac{1}{\vartheta} - \frac{1}{\vartheta_0} = A$$

where A is for most practical purposes a constant.

Usually the instrument is furnished with a table showing the relation between the current through the lamp and the temperature both with and without the absorption glass. If, however, this relation is not given when the absorption glass is used, it may be readily determined by measuring the constant A in the above formula. To do this, sight without the absorption glass on a muffle furnace or any uniformly heated furnace at 1,200 to 1,500°C. and observe the temperature ϑ, in degrees absolute. Then with the absorption glass in place, match the filament again and observe to what temperature, ϑ_0, in degrees absolute the current through the lamp corresponds. The difference in the reciprocal of these two temperatures is the constant A, which is usually of the order of magnitude -0.0002. This determination should be repeated several times and at several different temperatures of the furnace. The separate values of A should not differ by more than 1 per cent and the mean value is used for computing the relation between the observed absolute temperature with the absorption glass and the true temperature of the source. In making these computations care must be exercised that all temperatures are converted to degrees absolute. The following table illustrates the calibration of a certain pyrometer, both with and without the absorption glass. The constant of the absorption glass has the value $A = 0.000280$. By use of this glass temperatures as high as 2,730°C. can be measured although the temperature of the lamp does not exceed 1,360°C.

TABLE 11

Current, amperes	Temperature, degrees Centigrade	
	Without absorption glass	With absorption glass
0.26	634	943
0.28	765	1,190
0.30	860	1,386
0.32	936	1,555
0.34	1,002	1,710
0.36	1,060	1,854
0.38	1,113	1,992
0.42	1,201	2,237
0.46	1,281	2,478
0.50	1,359	2,733

It will be noted that the range of currents required is small, from in general about 0.3 to 0.6 amp. Thus if the ammeter is designed to give full-scale deflection on 0.6 amp. nearly half of the scale, from 0 to 0.26 amp. is never used. The Hickok depressed zero ammeter now furnished with the L. and N. pyrometer meets this objection. The moving-coil system including the supports, pivots and pointer may be adjusted relative to the magnet, by turning a lever on the case of the instrument, to one of two positions. In one position the pointer is adjusted *on open circuit* so that it falls over the first graduation on the scale. This adjustment is similar to the ordinary zero adjustment on any ammeter. In the second and working position the zero is de-

pressed off the scale an amount equivalent to 0.26 amp. The lowest scale reading is accordingly 0.26 and the entire scale from 0.26 to 0.60 amp. is thus utilized for the range of the pyrometer lamp.

General Use of Optical Pyrometers.—Optical pyrometers and radiation pyrometers, described later, afford the only means yet developed for measuring temperature above 1,500°C. The high temperature scale above 1,500°C. is based on the extrapolation of Wien's radiation law by means of a pyrometer of the Leeds & Northrup type. When the instrument is especially designed for precision work it is possible to measure a temperature difference of 0.2°C. at 1,500°C. The commercial form of the instrument when properly calibrated can be relied upon to 5°C. With a well-designed optical pyrometer there is a perfect color match of the two fields at all times. Hence, contrary to the general impression, color is not matched at all but simply brightness of uniform color. A color-blind observer will obtain the same settings as a normal observer. Forsythe[1] has compiled data obtained with an optical pyrometer of the Leeds & Northrup type by six observers none of whom had ever used an optical pyrometer before. The average variation from the mean for the six inexperienced observers was 3°C. and the maximum variation, 5°C.

Although the optical pyrometer is essential for the measurement of temperatures above 1,500°C. its usefulness is by no means confined to the high temperature range. The thermocouple cannot be adapted to many processes at low temperatures for example, the measurement of the temperature of steel rails as they pass through the rolls, ingots and forgings in the open and small sources such as a heated wire or lamp filament. The temperatures used in the above processes may be accurately measured by the optical pyrometer. The temperature of a microscopic sample of any material can be measured by a modified form of the Leeds & Northrup pyrometer.[2] Also in many processes a thermocouple is not so convenient to use as an optical pyrometer, especially when the temperature is not required often enough to warrant a permanent installation of thermocouples.

One serious objection to the optical pyrometer from the industrial point of view is the fact that it has not been made automatically recording. Since a photometric match is required for every setting the instrument necessitates the attention of an observer, although a satisfactory automatic device will be developed eventually. Another objection is the introduction of the human element into the readings, thus affording an opportunity for dishonest or prejudiced settings. The observer, if he is the operator of the furnace, should be taught that the instrument is for his own assistance and that it is not to be considered as a policial measure. Otherwise the measurements should be made by a disinterested party. In a plant operating several furnaces an intelligent boy can be profitably employed whose sole work is to make the rounds of the various furnaces and measure and record the temperatures.

Black-body and Non-black-body Conditions.—Optical pyrometers are usually calibrated to read correctly when sighted on a black body. Many furnaces approximate black-body conditions very satisfactorily. In a perfect black body the details of the inside of the furnace vanish and a piece of steel, for example, which is being heated cannot be distinguished from the back ground. If the

[1] *Gen. Elec. Rev.*, Sept., 1904, p. 753.
[2] BURGESS, Bureau of Standards *Sci. Paper* No. 198.

objects in the furnace can be distinguished but only on close observation, and if much of the detail is lost, after *they have been in the furnace some time*, it is not likely that the temperature measurement will be seriously in error. If in error at all, the observed temperature will be too high when the furnace walls are brighter than the material being heat treated and too low when the walls are less bright. This latter condition is possible if the heat supply is variable or if it is shut off and the furnace allowed to cool.

That a steel ingot placed in a heated furnace may appear much hotter than it really is, is a fact not always appreciated. The surface of the ingot appears hot because it reflects the bright light from the walls of the furnace. Of course in comparison to the much greater brightness of the walls, the cold ingot appears black, but this is due to the intense contrast. If the ingot is viewed alone when the *direct* radiation from the furnace is screened from the eye, it also is bright. Thus when an optical pyrometer is sighted on an ingot in the furnace, part of the light reaching the instrument comes from the side walls and is reflected by the surface of the ingot. Iron oxide reflects about the least amount of any material met with in metallurgical practice. Its emissivity is approximately 0.95; hence its reflection coefficient is 0.05. Suppose an iron ingot at room temperature were suddenly placed in a furnace at 1,200°C. Although the surface of the ingot is cold it reflects 5 per cent of the light falling upon it from the hot side walls. This 5 per cent of reflected radiation gives the ingot the appearance of an object at 950°C. The measurement by the optical pyrometer accordingly would be 950°C., although actually the ingot is at room temperature. The higher the reflecting power of the material the greater the observed temperature under the above conditions. Thus cold platinum would appear to be at about 1,160° or at almost the same temperature as that of the furnace. One method for reducing the error due to reflected radiation is to view the object through a large open door on a surface parallel to the opening. If the door is large enough the surface of the ingot thus loses a great portion of the furnace wall contributing the reflected light. The ingot reflects the image of the opening left by the door and this opening is not radiating. It must be pointed out that the reflection from any material being heat treated is diffuse like that from a matt surface so that an opening in a furnace wall is not truly imaged as would be the case with a polished mirror surface. Stray reflection is, however, greatly minimized even with matt surfaces by sighting through a large door on a surface parallel to the opening. When the material has attained the temperature of the side walls it is of course not desirable to open a large door since then the opening affects the black-body conditions. The pyrometer should be sighted through a small peep hole as soon as approximate temperature uniformity is obtained. A more satisfactory method for reducing the stray reflections than by opening a large door is to sight into a deep wedge-shaped cavity or hole made in the metal being heat treated. If this cavity is deep enough, very little radiation from the side walls can be reflected from it. If such a hole cannot be made conveniently, a length of iron pipe closed at one end, or a porcelain tube, may be placed on the material and so aligned that the pyrometer may be sighted through a peep hole directly into the bottom of the tube.

The effect of reflected light is very noticeable in an empty coke oven. The reflection coefficient of the brick walls is comparatively high. If a perfect mirror were placed parallel to a glowing surface the mirror would appear as hot as the surface. This effect of reflection takes place in a coke oven so that both walls appear of approximately equal brightness even though they may differ considerably in temperature. Frequently a patch of the wall on one side becomes coated with a layer of coke. Since the coke has a higher emissive power than brick this patch appears much hotter.

Actually it is at about the same temperature as the less bright surrounding wall. On account of reflection a corresponding bright patch appears on the opposite wall although this wall may be free from coke. It is evident that the measurement of temperature of a portion of a non-uniformly heated furnace by means of an optical pyrometer is difficult unless the precautions suggested above are taken. As soon as the furnace attains temperature uniformity and equilibrium the optical pyrometer gives the true temperature very easily and readily.

When an optical pyrometer is sighted on a glowing material in the open it reads too low. Certain materials, important industrially, have a very high emissivity so that the corrections necessary to add to the observed temperatures to convert them into true temperatures are small. Thus with iron oxide the correction is only 10° at 1,200°C. The corrections are very large for clear molten metals, but are smaller for the oxides which soon form on the molten surface when exposed to the air. Table 12 shows the true temperatures corresponding to the temperatures observed when sighting on certain materials in the open. For temperature control it is unnecessary to apply these corrections. The observed temperatures, although known to be low, will be low by the same amount from time to time and hence will serve just as satisfac-

TABLE 12.—TRUE TEMPERATURES *vs.* APPARENT TEMPERATURES MEASURED BY OPTICAL PYROMETERS USING RED LIGHT ($\lambda = 0.65\mu$) WHEN SIGHTED UPON THE FOLLOWING MATERIALS IN THE OPEN

Observed temperature, degrees Centigrade	True temperature, degrees Centigrade						
	Molten copper	Molten iron[1]	Solid iron oxide	Solid nickel oxide	Nichrome or chromel	Molten slag[2]	Bright platinum
700	.....	.....	700	701	702	.....	750
800	.....	.....	801	802	804	.....	861
900	.....	.....	902	904	906	.....	973
950	1,088	.....	953	955	958	.....	1,030
1,000	1,150	.....	1,004	1,007	1,010	.....	1,087
1,050	1,213	.....	1,055	1,058	1,063	.....	1,144
1,100	1,277	1,183	1,106	1,110	1,116	.....	1,202
1,150	1,341	1,239	1,158	1,162	1,170	.....	1,260
1,200	1,405	1,296	1,210	1,215	1,224	.....	1,320
1,250	1,470	1,353	.....	1,267	.....	.....	1,375
1,300	1,536	1,410	.....	1,320	.....	.....	1,435
1,400	.....	1,525	.....	....	.....	1,455	1,555
1,500	.....	1,641	.....	.....	.....	1,565	1,675
1,600	.....	1,758	.....	.....	.....	1,670	
1,700	.....	1,876	.....	.....	.....	1,780	
1,750	.....	1,935	.....	.....	.....	1,830	

[1] Computed for $E\lambda = 0.40$, this being the best value for ordinary steel practice.
[2] Computed for $E\lambda = 0.65$, an average value for liquid slags.

torily for reproducing temperature conditions in any process as the corrected temperatures. The above statement must be modified if factors other than emissivity of the material require consideration. For example, reproducible results cannot be

expected if heavy clouds of smoke are in the line of sight one day and not on the next day. If the pyrometer is sighted on a stream of molten iron during pouring or tapping the surface of the metal is usually clear and free from oxide. If the stream should at any time contain much slag, the surface will show bright patches on account of the higher emissivity of the slag. To make the readings conform with those taken on the clear stream, one must sight on the darker spaces between the slag or sight upon the slag and correct both sets of data according to Table 12.

This table was computed from the following equation where ϑ is the true absolute temperature, S the observed absolute temperature and $E\lambda$ the emissivity for the wave length λ. This wave length has been selected as $\lambda = 0.65\mu$, the approximate value for optical pyrometers.

$$\frac{1}{\vartheta} - \frac{1}{S} = \frac{\lambda \log E\lambda}{6{,}232} = \frac{\log E\lambda}{9{,}588}$$

The following table gives the emissivity of various materials for this wave length. The change of emissivity with temperature is usually small for metals.

TABLE 13.—MONOCHROMATIC EMISSIVITY FOR RED LIGHT (λ = ABOUT 0.65μ)

Material	$E_\lambda = 0.65\mu$	Material	$E_\lambda = 0.65\mu$
Silver	0.07	Cuprous oxide	0.70
Gold, solid	0.13	Iron oxide, 800°C	0.98
Gold, liquid	0.22	Iron oxide, 1,000°C	0.95
		Iron oxide, 1,200°C	0.92
Platinum, solid	0.33	Nickel oxide, 800°C	0.96
Platinum, liquid	0.38	Nickel oxide, 1,300°C	0.85
Palladium, solid	0.33		
Palladium, liquid	0.37		
Copper, solid	0.11	Iron (solid and liquid)	0.37
Copper, liquid	0.15	Nickel (solid and liquid)	0.36
Tantalum, 1,100°C	0.60	Iridium	0.30
Tantalum, 2,600°C	0.48	Rhodium	0.30
Tungsten, 1,000°C	0.46	Graphite powder (estimated)	0.95
Tungsten, 2,000°C	0.43		
Tungsten, 3,000°C	0.41	Carbon	0.85
Nichrome, 600°C	0.95		
Nichrome, 900°C	0.90	Porcelain (??)	0.25 to 0.5
Nichrome, 1,200°C	0.80		

Table 14 shows the corrections which must be added to the readings obtained with an optical pyrometer using light of wave length $\lambda = 0.65\mu$, for various emissivities, in order to obtain the true temperatures. These data are especially useful when carefully plotted with observed temperatures as abscissæ and corrections as ordinates. A family of curves is thus obtained corresponding to the different values of the emissivity.

TABLE 14.—CORRECTIONS TO OBSERVED TEMPERATURES FOR PYROMETER USING RED LIGHT
($\lambda = 0.65\mu$, $C_2 = 14{,}350$)

Emissivity	Add corrections below for the following observed temperatures, degrees Centigrade										
	700	800	900	1,000	1,100	1,200	1,300	1,400	1,600	1,800	2,000
0.30	55	67	80	95	111	129	148	168	213	264	322
0.40	41	50	60	71	83	96	110	125	158	195	237
0.50	31	37	45	53	62	71	82	93	117	144	175
0.60	22	27	33	39	45	52	59	67	85	104	126
0.70	16	19	23	27	31	36	41	47	59	72	87
0.80	10	12	14	17	19	22	25	29	36	44	54
0.90	5	6	7	8	9	10	12	14	17	21	25
1.00	0	0	0	0	0	0	0	0	0	0	0

Temperature of Glowing Gauze.—An interesting application of the optical pyrometer is for the measurement of the temperature of gauze electrically or otherwise heated. In certain chemical processes, platinum gauze electrically heated is used as a catalyzing agent, and must be maintained at a constant temperature. This is readily done by sighting normally on the surface of the gauze with an optical pyrometer. The observed temperatures may be thus exactly reproduced from day to day. If it is required to convert the observed temperatures into exact true temperatures of the wire forming the gauze the problem is difficult. An approximate solution satisfactory for all industrial work is, however, easily obtained.

We will assume that the mesh of the gauze is sufficiently coarse that multiple reflection between the separate wires is negligible. Let A_1 = the fractional part of the total area of the gauze comprised by the wire and A_2 = the fractional part of the total area representing the space between the wires. Let E_λ = the emissivity of the metal employed and E'_λ = the effective emissivity of the gauze as a whole, that is, taking into consideration the spaces between the wires which of course are not radiating surfaces. The following equations are readily apparent:

$$E_\lambda' = \frac{A_1}{A_1 + A_2} E_\lambda = A_1 E_\lambda \text{ since } A_1 + A_2 = 1$$

$$\frac{1}{\vartheta} - \frac{1}{S} = \frac{\log A_1 E_\lambda}{9{,}588}$$

where ϑ is the true absolute temperature of the wire of the gauze and S is the absolute temperature observed with an optical pyrometer sighted normal to the surface.

A platinum gauze commonly employed is number 80-mesh (80 wires to the inch) of 0.003-in. wire. For this gauze $A_1 = 0.42$. The emissivity of bright clean platinum is 0.33. The platinum of this gauze soon becomes somewhat corroded. Possibly an emissivity of say 0.4 is more nearly the correct value under these conditions. Hence, the effective emissivity of the gauze $= A_1 E_\lambda = (0.42)(0.4) = 0.17$. Usually the gauze must be viewed through a glass window. A thin glass window (see below) transmits about 90 per cent of the light falling upon it. Hence, the final effective emissivity, using a glass window, $= E'_\lambda = (0.17)(0.90) = 0.15$. The following table was computed by the formula $1/\vartheta - 1/S = \log 0.15/9{,}588$. A similar table for other gauzes may be computed in the manner outlined.

TABLE 15.—PLATINUM GAUZE, 80-MESH, 0.003-IN. WIDE. TEMPERATURES OBSERVED THROUGH ONE WINDOW BY OPTICAL PYROMETER SIGHTED NORMAL TO SURFACE OF GAUZE, *vs.* TRUE TEMPERATURE OF GAUZE

Observed temperature, degrees Centigrade	True temperature, degrees Centigrade	Observed temperature, degrees Centigrade	True temperature, degrees Centigrade
600	675	850	975
650	730	900	1,035
700	790	950	1,095
750	850	1,000	1,160
800	910	1,050	1,220

Use of a Window.—It is frequently necessary, especially in the laboratory, to sight an optical pyrometer into a furnace through a window. What correction must be applied to the observed temperatures to take account of the loss of light at the window? Kanolt has measured the transmission coefficient for a number of ordinary glass windows at $\lambda = 0.65\mu$, and obtained a mean value of 0.904. Hence we have

$$\frac{1}{\vartheta} - \frac{1}{S} = \frac{\log 0.904}{9,588} = -0.0000046$$

where ϑ is the true absolute temperature of the source and S is the observed absolute temperature. The following table is computed from the above formula:

TABLE 16.—CORRECTION TO OBSERVED TEMPERATURES FOR ABSORPTION OF LIGHT BY A SINGLE CLEAN WINDOW

Observed temperatures, degrees Centigrade	Correction to add, degrees Centigrade	Observed temperatures, degrees Centigrade	Correction to add, degrees Centigrade
600	3.5	1,600	16.0
800	5.4	1,800	20.0
1,000	8.0	2,000	24.0
1,200	10.0	2,500	36.0
1,400	13.0	3,000	50.0

Flames and Smoke.—The optical pyrometer cannot be used satisfactorily when sighted through flames or smoke. Usually the presence of dense flames increases the temperature reading, and the presence of smoke clouds absorbs so much radiation that the pyrometer may read several hundred degrees low. The optical pyrometer can be used to measure the temperature of the slag in an open-hearth furnace but the flames prove a serious hindrance except during reversals when observations may be taken to advantage. In a cement kiln the dust, smoke and flames all combine to make the observations very untrustworthy. Carbon dioxide, water vapor and other invisible gases produce no effect.

In many processes where smoke cannot be eliminated or where black-body conditions are not satisfactory, a porcelain or other refractory tube with a closed end is inserted into the furnace. The pyrometer is sighted into this tube which if fairly uniformly heated over a sufficient area affords an excellent black body. This method has been employed also for obtaining the true temperature of molten metals but suitable refractory tubes for many molten metals have yet to be developed.

Radiation Pyrometry.—An optical pyrometer measures the intensity of a narrow spectral band of radiation emitted by a glowing object. The radiation pyrometer measures the intensity of all wave lengths, the light rays and the heat rays combined. Usually the energy of all wave lengths radiated by the source is focused in some manner upon the hot junction of a small thermocouple. The temperature to which this junction rises is approximately proportional to the rate at which energy falls upon it, which in turn, by the Stefan-Boltzmann law, is proportional to the fourth power of the absolute temperature of the source. The rise in temperature of the hot junction of the couple generates a thermoelectric electromotive force. Hence, the calibration of a radiation pyrometer consists in determining the relation between the electromotive force developed and the temperature of the source sighted upon. This relation follows the law $e = a\vartheta^b$ where ϑ is the absolute temperature of the source, e is the electromotive force developed by the instrument and a and b are empirical constants determinable by two standardization points. The electromotive force may be measured by a potentiometer or galvanometer, or by any of the methods discussed under thermoelectric pyrometry. The galvanometer should have as high a resistance

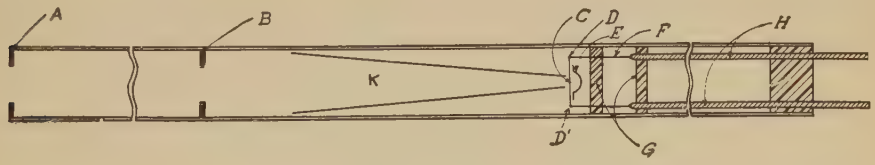


FIG. 21.—Thwing radiation pyrometer.

as is consistent with the requirement of robustness. The same type of instrument is used with the radiation pyrometer as with the ordinary thermocouple. In fact the entire discussion on galvanometric methods of measuring electromotive force of thermocouples is equally applicable to radiation pyrometry. The temperature of the cold junction of the couple in the radiation pyrometer is not controlled. The hot and cold junctions are in fairly close proximity and are hence equally affected by changes in room temperature. The cold junction is always shaded from the heat radiated by the source sighted upon.

Figure 21 illustrates the principle of the Thwing radiation pyrometer made by the Thwing Instrument Co. Radiation from the furnace enters the diaphragm A and falls upon the hollow conical mirror K. The hot junction C of a minute thermocouple is located at the apex of the cone and the cold junctions are at D and D'. By multiple reflection along the sides of the conical mirror the radiation is finally concentrated upon the hot junction of the couple. The electromotive force is measured by a galvanometer graduated to read temperature directly. Except for incidental errors which will be considered later, the reading of the instrument is independent of the sighting distance provided the diameter of the source is sufficient to fill the cone of rays defined by the geometrical construction of the receiving tube. The

Thwing instrument is so constructed that the source must have a diameter at least one-eighth of the distance from the source to the receiving tube. Thus at 8 ft. from a furnace the opening into the furnace must be 1 ft. in diameter. For permanent installations the tube is ventilated and has several extra diaphragms to prevent local heating of the instrument and reradiation to the couple.

Figure 22 illustrates the principle of the Foster radiation pyrometer made by the Taylor Instrument Cos. The thermocouple b and a front diaphragm B are located

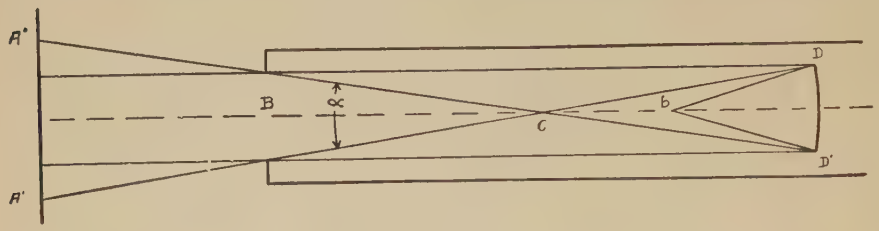


FIG. 22.—Foster radiation pyrometer.

at the conjugate foci of a concave mirror DD'. As in the case of the Thwing pyrometer, the source must be large enough to fill the cone of rays defined by the angle α or the lines $A'CA''$. The position of the point C is marked by a wing nut on the telescope tube. The angle α is made such that the diameter of the source sighted upon must be at least one-tenth the distance from the source to the wing nut. Thus for a

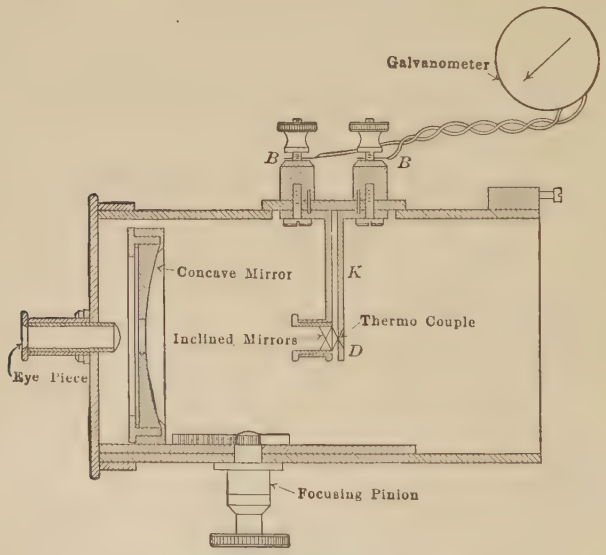


FIG. 23.—Fery radiation pyrometer.

distance of 10 ft. the diameter of the opening into the furnace must be at least 1 ft. The Brown Instrument Co. makes a radiation pyrometer which is similar in principle to the Foster pyrometer. The receiving tube of the Brown radiation pyrometer is made collapsible for convenience in carrying.

Figure 23 is a cross-section drawing of the Fery pyrometer made by the Taylor Instrument Cos. Radiation from the source sighted upon is concentrated by the concave mirror of speculum metal or gold, upon the hot junction of a minute thermo-

couple. This instrument is not fixed focus as is the case with the other types of radiation pyrometer but requires focusing for each sighting distance. The focusing is simply accomplished by means of an ingenious device due to Fery. Two semi-circular mirrors Fig. 24 (*a*) inclined to one another at an angle of 5 to 10° are mounted in the thermocouple box, an opening of about 1.5 mm. at the center of the mirrors forming the limiting diaphragm immediately in front of the couple. The observer views by means of the telescope *D* the image of the furnace formed by the large concave mirror

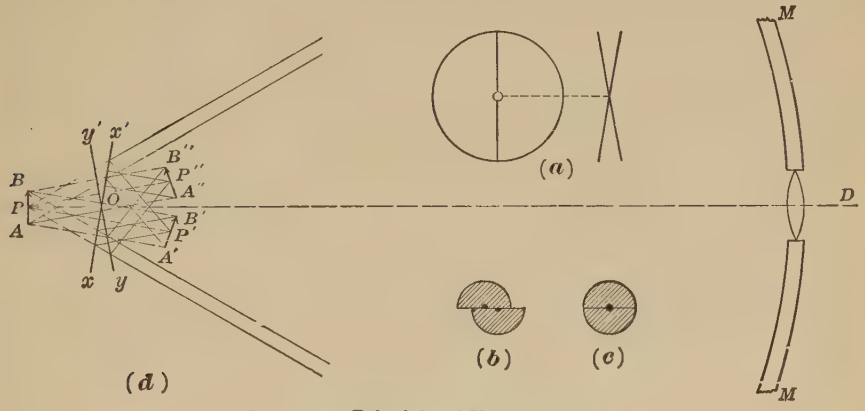


FIG. 24.—Principle of Fery pyrometer.

M,M and reflected by the inclined mirrors xx' and yy' through a hole in the large mirror. If the image is not correctly focused at *O*, the intersection of the two small mirrors, the image appears broken in half as shown by (*b*). Correct focus is obtained when the two halves of the image are in alignment (*c*). Thus until correct focus is obtained certain straight lines of the source appear broken in the image. This breaking of a line is illustrated on a magnified and distorted scale by (*d*). Suppose that the pyrometer were incorrectly focused upon a line source, an arrow, the image falling at position *AB* instead of at *O*. The image of the arrow reflected from the mirror *YY'* lies at *A''B''* and that reflected from the mirror xx' at *A'B'*. To the observer at *D* the projections of these images appear as two distinct arrows thus $\rightarrow\ \rightarrow$. As the pyrometer is brought nearer into the correct focus by turning the pinion screw and thus moving the large concave mirror in the direction *OD* the points *P'* and *P''* of the reflected images move along the lines *P'O* and *P''O* coinciding at *O* when the correct focus is obtained. The two arrows are then superposed forming a single image.

For the measurement of very high temperatures, usually above 1,500°C., the cover to the front of the telescope is provided with a sectored opening which may be adjusted to reduce the radiation falling upon the receiver by any definite amount, and in this manner the upper temperature range of the instrument is practically unlimited. This adjustment is made by the manufacturer and should not be altered. For the lower scale range the cover is open.

The readings with a Fery pyrometer when properly focused, neglecting secondary errors discussed later, are independent of the sighting distance, as is the case with the fixed focus radiation pyrometer.[1] The image of the source, as viewed through the small telescope, must cover *completely* the limiting diaphragm to the thermocouple. This diaphragm appears as a black circular area shown at the center of the field in Fig. 24 (*b*) and (*c*). An excellent rule to follow is to sight at such a distance that the

[1] See Bureau of Standards *Sci. Paper* No. 250, p. 97, for the geometrical demonstration of this fact.

area of the image overlaps this hole and extends half way to the edge of the focusing mirrors. The Fery pyrometer requires a smaller source than the fixed-focus instruments. The following table illustrates the size of source required for various sighting distances in order that the image cover the limiting diaphragm and extend about half way to the edge of the focusing mirrors, *i.e.*, diameter of image = 4 mm.

TABLE 17

Sighting Distance, Centimeters	Diameter of Source Centimeters
70	3.2
80	3.7
100	4.8
150	8.3
200	11.2
300	16.8
500	28.5

Errors to Which Radiation Pyrometers are Subject.—Dust and dirt allowed to accumulate upon the concave reflecting mirror may so decrease its reflection coefficient as to develop errors amounting to 100° or even 200°C. Frequently the dust can be removed from the mirror by carefully brushing with a camel's hair brush. The mirror may be removed from the instrument and washed, but this must be done with great care to insure that the delicate thermocouple or its mounting is not disturbed. The safest practice is to take all possible precautions to prevent dust from entering the instrument. Keep the case closed, or the front diaphragm of the fixed focus instruments plugged with a cork when not in use.

As shown above, the simple theory of the radiation pyrometer states that the readings are independent of the sighting distances or size of source provided the source is larger than the minimum size of source demanded by the geometry of the instrument. Actually this ideal condition is not always realized. Some stray radiation is reflected down the walls of the telescope case. The walls become heated by the furnace and re-radiate to the couple. Limiting diaphragms are similarly heated and re-radiate to the couple. For these reasons a radiation pyrometer tends to read low the greater the sighting distance or the smaller the size of source. It is thus in general desirable to use a radiation pyrometer as nearly as possible in the same manner from day to day and to have it specially calibrated for such use. For example, if a 6-in. source is employed, decide upon a convenient sighting distance, say 4 ft., and always use this distance, and not sometimes 1 ft. and sometimes 5 ft. In the Fery pyrometer both the proper size of source and the proper focusing distances are secured by following the rule suggested above, *viz.* focus at such a distance that the image extends half way between the black hole and the outside edge of the focusing mirrors.

Advantages and Disadvantages of Radiation Pyrometers.—For temperatures above 1,400 or 1,500°C. either a radiation pyrometer or an optical pyrometer must be employed. The optical pyrometer is capable of higher accuracy and is less susceptible to errors than the radiation pyrometer. Smoke and dust affect the readings of both instruments but the radiation pyrometer is affected by the presence of cooler strata of carbon dioxide and other gaseous combustion products in the furnace. Carbon dioxide and water vapor absorb the heat rays and hence the radiation pyrometer will read too low when sighted through such gases. The main advantage of the radiation pyrometer is the fact that it can be made automatically recording. The recording mechanism is the same as that employed for ordinary thermocouples, discussed in the section on recording pyrometry. The radiation pyrometer is desirable for many processes of lower temperature where thermocouples cannot be conveniently installed. It is also useful in measuring the surface temperature of large ingots.

Black-body and Non-black-body Conditions.—Radiation pyrometers are calibrated to read correctly when sighted upon a black body. Most furnaces approximate black-body conditions sufficiently well. When sighted on materials in the open, certain corrections must be applied to the observed temperature to convert them into true temperatures. In case the temperature of one section of an unequally heated furnace is required or if the furnace contains much smoke or dust it is frequently desirable to sight the radiation pyrometer into the bottom of a fireclay or porcelain tube as illustrated by Fig. 25. The tube should be uniformly heated at the end for a length at least three times its diameter. The tube must have a diameter such that the cone of rays entering the telescope is not intercepted by the front of the tube. Figure 26 illustrates this condition. Suppose the distance from the bottom of the tube to the mirror of the Fery pyrometer is 100 cm. Referring to Table 17, the diameter of the source required is 4.8 cm. Lay off the distance b = 4.8 cm. and draw straight lines from the bottom of b to the bottom of the mirror c, and from the top of b to the top of the mirror. The tube must have such a diameter at its front end a that it does not cut in on the cone of rays represented by these two straight lines. This can be determined only by actually making the above drawing to scale or by computing. It can not be decided by looking through the

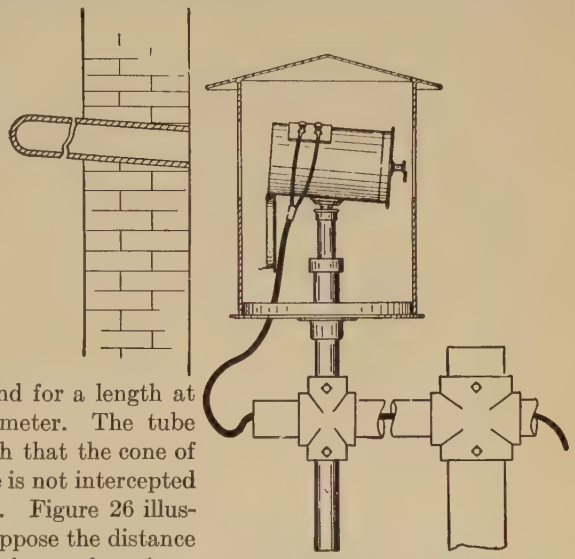


FIG. 25.—Use of porcelain tube for observing temperature.

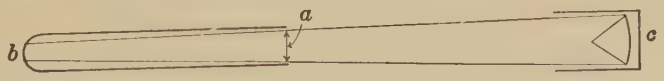


FIG. 26.—Porcelain tube for observing temperatures.

telescope of the pyrometer since the front end of the tube would not necessarily show even if it did cut in on the cone of rays. In the case of the fixed-focus instruments the diameter of the tube must be such that the cone of rays cuts the tube in the region which is uniformly heated.

The following table shows the true temperatures corresponding to the apparent or measured temperatures observed with a radiation pyrometer when sighted upon various materials in the open. This table must not be confused with Table 12 for the optical pyrometer. It will be noticed that the corrections are entirely different for the two types of pyrometer.

TABLE 18.—TRUE *vs.* APPARENT TEMPERATURES MEASURED BY RADIATION PYROMETERS WHEN SIGHTED UPOM MATERIALS IN THE OPEN

Observed temperature, degrees Centigrade	True temperature, degrees Centigrade				
	Molten iron	Molten copper	Copper oxide	Iron oxide	Nickel oxide
600	.....	1,130	720	630	710
650	.....	1,210	775	.....	755
700	.....	1,290	830	735	800
750	.....	.....	890	.....	845
800	1,200	.....	945	840	895
850	1,270	.....	1,000	.....	940
900	1,340	.....	1,060	945	985
950	1,410	.....	1,115	.....	1,030
1,000	1,475	.....	1,170	1,050	1,075
1,050	1,550	.....	.....	.....	1,120
1,100	1,610	.....	.....	1,155	1,165
1,150	1,680	.....	.....	.....	1,210
1,200	1,750	.....	.....	1,260	1,255

Resistance Thermometry.—The operation of a resistance thermometer depends upon the variation with temperature of the resistance (mass resistivity) of an electrical conductor. A high temperature coefficient of resistivity might accordingly appear of primary importance. Certain oxides possess this property, having a resistance at room temperature from a hundred thousand to a million times that at 1,000°C. However, the resistance of a suitably mounted material may be easily and accurately determined so that the consideration of high temperature coefficient is rarely, if ever, the deciding factor in the choice of the material. The fundamentally important property is reproducibility and constancy of resistance. A further desirable feature is a simple and convenient relation between temperature and resistance. Platinum of the highest obtainable purity meets these requirements up to about 1,100°C. The temperature coefficient of resistance of platinum is so well established that the working temperature scale up to 1,083°C. is based upon this form of pyrometer calibrated according to certain specifications. On the basis of this reproducible working temperature scale, temperatures may be measured to about 0.1°C. at 1,000°C. To obtain such precision, however, requires accurately constructed apparatus and very careful experimental manipulation.

For industrial purposes, resistance thermometers are made usually either of platinum or of nickel, the latter material being fairly satisfactory for temperatures below 300°C. In the form of platinum thermometer made by Leeds & Northrup Co. about 40 cm. of fine wire is wound upon a mica frame into a spiral coil about 4 cm. in length and 0.7 cm. in diameter. The length of the wire is so adjusted that the resistance of the coil is about 8.3 ohms at 0°C. The frame is made of crossed strips of thin mica notched at the edges to hold the wire in place. From the ends of this coil lead wires of platinum or gold are carried to the terminal head of the thermometer. The lead wires are insulated and held apart by mica discs through which the wires are threaded.

The thermometer is protected by a porcelain tube glazed on the outside, or by a quartz tube which fits into the terminal head. For industrial use the refractory tube is itself protected by an outer metal tube. The metal tube is usually fitted to an extra terminal head through which the lead wires to the line are carried. On account of the deterioration of the platinum, thermometers constructed of fine wire are not very satisfactory for industrial use above 900°C. Heavy platinum wire, 0.6 mm. in diameter, may be employed up to 1,100°C. but the resistance of the thermometer is so low that it is necessary to use methods of measurement which do not depend upon constancy in the resistance of the lead wires, *e.g.* Thomson bridge or potentiometer.

In the Engelhard resistance thermometer a coil of fine platinum wire having a resistance of 50 ohms at 0°C. is wound upon a quartz rod 3 mm. in diameter and 6 cm. in length and fused inside a quartz tube so that the platinum is entirely embedded in quartz. Gold leads are carried from the coil to the terminal head. The quartz insures high electrical insulation while still preserving excellent heat conduction. The thermometer is compact and the platinum is thoroughly protected against deterioration. For work of the highest precision, in which measurements to 0.1° or 0.01° are required this type of thermometer is inferior to that in which the platinum is loosely mounted on a mica frame since its calibration is slightly altered with use on account of the strain introduced by the differential expansion of quartz and platinum. For most purposes this objection is of small consequence.

The relation between the temperature t°C. and the resistance r_t of the platinum resistance thermometer is of the following form where a, b, and c are constants determinable by the standardization points.

$$r_t = a + bt + ct^2$$

If only two lead wires are carried from the thermometer coil to the indicator, the resistance measured is the sum of the resistances of the coil, the platinum or gold lead wires to the head of the thermometer, and the copper lead wires from this point to the indicator. The resistance of the platinum or gold lead wires will depend upon the form of temperature gradient along the thermometer from the bulb to the head and upon the depth of immersion. Hence, this variable resistance is introduced into the temperature measurement, and changes in the resistance of the lead wires will be interpreted as changes in the temperature of the thermometer coil.

There are three general methods for minimizing or eliminating the effect of variation in lead resistance.

1. Use of a coil of high resistance: This is the method employed in the Engelhard thermometer. The resistance of the lead wires from the coil to the thermometer head and the likely variation in resistance of the copper lead wires are extremely small compared to the 50-ohm platinum coil. Hence such variations in resistance as are likely to occur do not introduce serious error. Thus only two lead wires are necessary, from the thermometer to the indicating instrument.

2. Compensating lead wires of the Callendar or Siemens form: Both of these methods of compensation are applicable to measurements made with the Wheatstone bridge. The Callendar form of compensation requires four lead wires. Two of the lead wires joined to each end of the coil are connected in one arm of the bridge and the other two lead wires, which are "dummy" leads formed of a single loop of wire extending to the top of the coil, are connected in the corresponding arm of the bridge. The two sets of lead wires are exactly alike so that variations in temperature affect each set similarly. The Siemens method of compensation described below requires three leads and is more often employed industrially in this country.

3. Potential terminals: The use of potential terminals is necessary when the resistance of the thermometer is made very low or when the precision required is such as to make variations in difference of lead resistances inadmissible.

The general methods for measuring resistance include the Wheatstone bridge, the Thomson bridge, potentiometer, and differential galvanometer. The Wheatstone bridge is used the most extensively for industrial installations.

Three-lead Wheatstone Bridge Method.—Figure 27 illustrates the wiring diagram for a simple Wheatstone bridge and thermometer with the Siemens three-lead compensation.

The platinum or gold lead wires $C'd$ and $T'e$ in the thermometer are constructed of as nearly the same resistance as possible, and the copper lead wires CC' and TT' must also have an equivalent resistance. The battery B is connected between the ratio arms r_1 and r_2 of the bridge and to the compensating lead wire cd. A sensitive

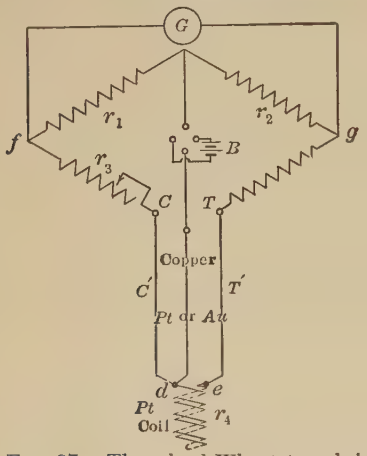


FIG. 27.—Three-lead Wheatstone bridge.

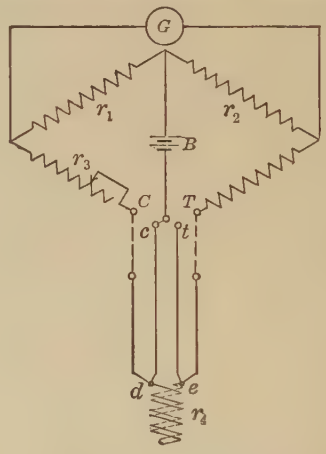


FIG. 28.—Four-lead Wheatstone.

galvanometer G is connected to the points f and g as illustrated. By varying the setting of the dial or slide wire r_3 the bridge is balanced until the galvanometer shows zero deflection. Whence from the principle of the balanced Wheatstone bridge we obtain the following relation.

$$\frac{r_3 + Cd}{r_1} = \frac{r_4 + Te}{r_2}$$

If r_1 is constructed equal to r_2, since $Cd = Te$, the setting of the slide wire or dial r_3 gives directly the value of the resistance of the coil, *i.e.* $r_3 = r_4$. Hence, the measured resistance is independent of the resistance of the lead wires. For industrial use the slide wire of the bridge is graduated directly in degrees of temperature.

Figure 28 illustrates the wiring diagram for a simple Wheatstone bridge and thermometer of the four-lead potential terminal type. This method is used mainly for precision measurements in the laboratory. The battery B is first connected to c. As before, when the bridge is balanced we obtain:

$$r_3 + Cd = r_4 + Te$$

The battery is then connected to the terminal t and the terminals T and C are interchanged. We accordingly have for a balance:

$$r'_3 + Te = r_4 + Cd.$$

Whence upon combining these two equations:

$$r_4 = (r_3 + r_3')/2$$

Thus the measured resistance of the coil is independent of the resistance of the lead wires and any error which would result from inequality of the lead resistances Cd and Te in the ordinary three-lead thermometer is eliminated.

Figure 29 illustrates the wiring diagram for a four-lead thermometer used with a potentiometer. The current adjusted to a suitable value by the rheostat r_1 flows through a resistance standard R and the thermometer coil r_4. The potential drops across R and r_4 are measured by the potentiometer. Whence from Ohm's law, $e/R = e'/r_4$ where e and e' are the potential drops across R and r_4 respectively. Thus knowing e, e' and R, the coil resistance r_4 is readily obtained. The resistance R may be made adjustable so that by varying R until $e = e'$ the dial setting gives directly the value of $r_4 = R$. This method is useful in precise laboratory work.

Deflection Galvanometer Method.—The bridge methods so far described require the adjustment of the bridge until the galvanometer shows zero deflection. The temperature is then read from the setting of the slide wire. It is possible,

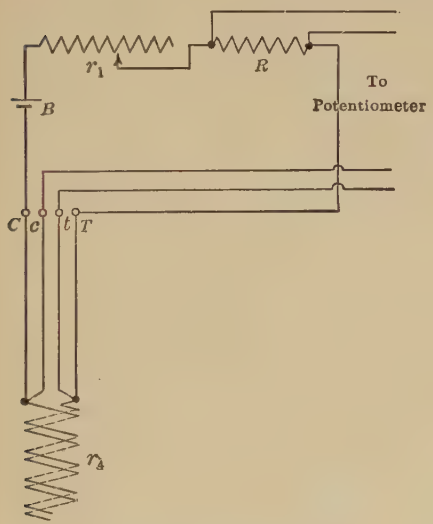


FIG. 29.—Four-lead potentiometer circuit.

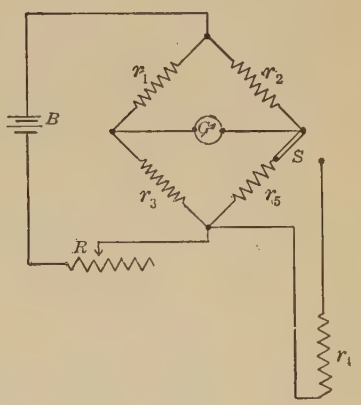


FIG. 30.—Unbalanced Engelhard Wheatstone bridge.

however, to use an unbalanced bridge with all resistances fixed, except that of the thermometer, which depends upon its temperature, and the temperature is obtained from the scale reading of the galvanometer. Figure 30 shows the wiring diagram for the Engelhard unbalanced Wheatstone bridge. With the switch S thrown to the right we have the simple Wheatstone bridge except that all the resistances r_1, r_2, and r_3 are fixed. At a certain temperature of the thermometer, for example $-10°C.$, its resistance is such that the bridge is balanced and $r_3/r_1 = r_4/r_2$. If the temperature of the thermometer increases, its resistance increases and the bridge is no longer balanced. Thus a current flows through the galvanometer producing a deflection. The deflection of the galvanometer in general depends upon two factors, the temperature of the thermometer and the current flowing from the battery B. This current is adjusted to a definite value in the following manner. The switch S is thrown to the left as illustrated, thus replacing the thermometer by a fixed resistance of such magnitude that a suitable value of the current from B deflects the galvanometer to its full-scale deflection or to a normal point marked on the scale. The resistance R is adjusted until the galvanometer deflects the proper amount.

The switch is then thrown to the right and since the factor of variable battery electromotive force or current is eliminated by this preliminary setting the scale of the galvanometer may be empirically graduated to read directly the temperature of the thermometer. Thus no further adjustments are necessary so long as this current remains steady, and any temperature is readily observed without the necessity for balancing required in the ordinary bridge. It is possible to have two or more temperature scales on the galvanometer. When two scales are provided, as for example 0 to 300°C. and 300 to 700°C., one of the arms of the bridge is made adjustable to either of two resistances, one giving zero deflection for a thermometer temperature of 0°C. and the other zero deflection for a temperature of 300°C.

Summary on Resistance Thermometry.—Although the resistance thermometer is of the greatest importance in the laboratory for work of extremely high precision its importance for industrial processes at high temperatures is waning. The precision of which the instrument is capable requires many precautions easily overlooked. Bad contacts in the line, a broken strand in a flexible cable, etc., may cause large errors. The thermometer is very fragile and its calibration may be altered by severe handling or by the slightest contamination or by volatilization of the platinum wire forming the coil. The range is confined usually below about 1,000°C., whereas rare-metal thermocouples may be used to 1,400°C.

In the early development of pyrometry before satisfactory galvanometers were devised for thermocouples, the resistance thermometer afforded about the only accurate means of temperature measurement, but with the highly sensitive and reliable thermocouple indicators and portable potentiometers and with the reproducible Le Chatelier couple now obtainable, there are few industrial processes carried out above 300°C. where a thermoelectric installation is not to be preferred. When very small temperature changes or intervals must be measured with high precision the resistance thermometer may be employed effectually. Also in certain processes where an integrated or average temperature over the entire bulb is required, the resistance thermometer is high desirable. A homogeneous thermocouple indicates the temperature of its hot junction while a resistance thermometer indicates the average temperature of the entire coil or bulb, which may be constructed as long as desired. There are no cold junction corrections for the resistance thermometer—a decided advantage over the thermocouple, but one which is offset by the disadvantages mentioned.

Recording Pyrometry.—The pyrometers which can be made to record automatically fall under the following classifications: (1) Gas, saturated vapor, and liquid thermometers; (2) resistance thermometers; (3) thermoelectric pyrometers; (4) radiation pyrometers.

Of these four types the thermoelectric pyrometer recorder has the greatest applicability, especially for the higher temperatures at which the first two named are not suitable. The constant-volume industrial gas thermometer is successful up to about 400°C. The resistance thermometer is capable of high accuracy up to 1,000°C. At such high temperatures, however, thermocouples are more serviceable since deterioration of the wire from continual heating does not so seriously alter the electromotive force developed by a couple as it does the resistance of a resistance thermometer. Radiation pyrometers are useful at the highest attainable temperatures, but processes in which extremely high temperatures are employed are not in general subjected to precise temperature control.

The most usual form of temperature record is that in which temperature appears as

one coördinate and time as the other coördinate. Such a temperature-time curve has been called an autographic record. This type of curve is the most easily obtained mechanically and is valuable as a continuous record of the temperature of a furnace over an extended interval of time. It may be employed to detect transformation points in steel, which appear as flexures or indentations on the plot when the furnace containing the sample of steel is uniformly heated or cooled. For this latter work, however, the "differential temperature" curve recorder is especially adapted and will be described later. The various special methods have been discussed by Burgess.[1]

A recorder for obtaining a temperature-time curve consists essentially of a deflectional instrument with a mechanism for periodically recording the position of the deflecting member upon a chart which moves with a uniform speed. As in the case of simple indicators there are two general types of recorders, one operating on the galvanometric principle, and one operating upon the potentiometric principle. Also, as in the case of indicating instruments the potentiometric principle while somewhat more complicated has the especially desirable feature that its indications are independent of the thermocouple resistance.[2]

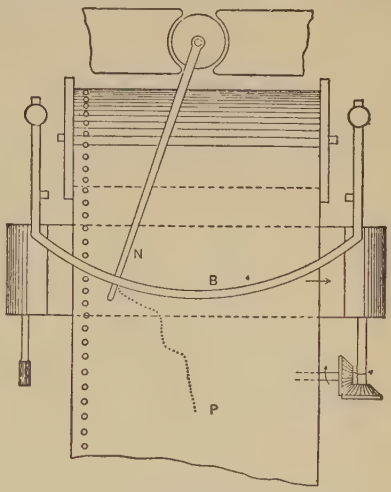


FIG. 31.--Recording device for thermocouple.

There are three types of record paper in general use, the roll charts, the drum charts and the disk or circular charts. The roll chart may contain enough paper to last a month or more, while usually the drum or circular charts are renewed every 24 hr. For single-point recorders all of these record forms are employed, but with one or two exceptions multiple-point recorders use record paper in the roll form. Upon circular charts the lines of equal temperature are represented by concentric circles and lines of equal time by arcs following the course of the galvanometer needle (see Fig. 31).

The drum and roll records are the best in point of available width of scale, and the latter is the most widely applicable for recording rapid changes of temperature, because the time coordinates on the drum chart and particularly on the disc chart are very short. The coordinates of the roll and drum charts can be made rectangular but in some cases the lines of equal time are portions of circles the radii of which are the length of the galvanometer needle. Thus in Fig. 31 the lines of equal times are arcs parallel to the chopper bar *B*.

The record may be obtained by pen and ink, stylus and inked ribbon, inked thread, carbon paper, or coated paper; by puncturing the paper by means of an electric spark; or by stamping upon the record sheet some imprint from a stencil. The various methods have been developed with a view to making more distinct the records of multiple point recorders.

The recording millivoltmeter or galvanometer may employ the same galvanometer system used in the portable indicating instruments. A more rugged instrument is desired, however, and is obtained by increasing the strength of the springs, boom, coil, and the pivots and bearings of pivot instruments. To compensate for the resulting decrease in sensitivity, the density of magnetic flux may be increased, the number of turns of wire on the moving coil may be increased, and the so-called swamping resis-

[1] Bureau of Standards, *Sci. Paper* No. 99.

[2] See paragraphs on Potentiometer, p. 426.

tance in series with the moving coil may be decreased. Because of the very small electromotive forces developed by thermocouples and the necessity of using a comparatively high resistance in series with the galvanometer coil to minimize the effect of a varying resistance, the torque which can be produced on the moving coil is small. The pointer cannot be used to trace a legible record directly since the friction between the pointer and paper would entirely alter the readings. One common method for obtaining the record is illustrated in Fig. 31. The paper is unwound from its roll, at uniform speed by clockwork. An inked ribbon lies below the paper and above a metal plate. At periodic intervals the chopper bar *B* falls pressing a stylus on the end of the galvanometer boom or needle *N* into contact with the paper and against the ribbon and metal plate underneath. This makes a small dot on the under side of the translucent paper which shows through from above as illustrated. The paper is ruled with the proper time and temperature coördinates and the row of dots obtained by continuous operation constitutes the required temperature-time curve. In general the dots may be made to appear at intervals of 10 to 60 seconds, depending on the natural period of vibration of the moving coil. The above principle is employed in many instruments of American make. The chopper bar may be operated by an electric motor, clockwork, or electromagnet, and the design must be such that the galvanometer pointer swings clear of the bar between intervals of depression, and such that the depression of the bar against the boom or pointer in no way damages the coil mounting. These conditions may be met in suspension, pivot, and combination pivot-suspension systems.

Industrial Types of Recorders.—The Circular-chart single-point recorders are made by Cleveland Instrument Co., the Bristol Co., Brown Instrument Co., and Hoskins Manufacturing Co.

Roll chart recorders are made by the Brown Instrument Co., Bristol Co., Charles Engelhard, Hoskins Manufacturing Co., S. H. Stupakoff, Taylor Instrument Cos., Thwing Instrument Cos., and Wilson-Maeulen Co. The potentiometer recorder of the Leeds & Northrup Co., which has a roll chart, is described in a later section.

General advantages of the roll chart are legibility (coördinates are parallel and may be made rectangular), width of scale, adaptability to rapidly changing temperatures, and the recording of more than one record on a single chart. With a paper speed of 1 in. per hour a 20-yd. roll will last 1 month. Rolls are generally supplied in lengths of 20 yds. or longer.

This type of recorder is easily made multiple recording for several couples by use of a commutating device for automatically switching successively into the circuit the various couples. The number of records which can be thus made on a single chart without confusion is usually four to six although occasionally as many as 12 records are employed.

Thermocouple Recorder Operating on the Potentiometric Principle.—The principle of operation of the potentiometer circuit and the various arrangements as it is applied in pyrometry has been described (see Fig. 6). The three steps in the operation, (1) adjusting the current in the battery circuit, (2) connecting the thermocouple and (3) moving the slide-wire contact until the galvanometer deflection is zero, are done automatically in the multiple-point potentiometer recorder, made by the Leeds & Northrup Co. In the curve-drawing recorder, which is a single-point instrument made by the same company, the battery current is adjusted by hand.

The potentiometer recorder consists essentially of a potentiometer with a mechanical movement for automatically changing the slide-wire contact and moving the pen or print wheel across the paper; also the parts necessary to move the paper

chart. Power is obtained from a small motor with an adjustable governor attached which will maintain the speed quite uniform in spite of large variations in the line voltage.

The chart of this instrument is 10 in. wide and has rectangular coördinates. The scale is uniform if graduated in millivolts and departs from uniformity according to the temperature electromotive force relation of the couple, if calibrated in degrees of temperature. In multiple-point recorders a print wheel carrying four or more numbers is substituted for the pen. This wheel is turned in synchronism with the commutator so that the proper number is printed. A small inked roller turns in contact with the print wheel. The latter is depressed once a minute, a length of time sufficient to allow the balancing action to be completed after each couple is connected.

The standard galvanometer is sufficiently sensitive to work satisfactorily with a full scale range of 10 millivolts which gives a very open scale particularly for base-metal couples, where $\frac{1}{10}$ in. of scale corresponds to $2\frac{1}{2}$°C. This recorder is also used with resistance thermometers in which case the electrical circuit takes the form of a Wheatstone bridge, and the scale can be opened until the entire range corresponds to so little as 2°C. The zero of the scale can be adjusted to correspond to any fixed electromotive force so that the scale may be put within any range of temperature desired. The great adaptability of the instrument is readily apparent.

Transformation Point Indicators and Recorders.—Such instruments are designed to indicate the temperature at which a chemical or physical transformation occurs. They are based on the fact that heat is absorbed or given off when such transformations take place. The simplest instrument will give a temperature-time curve of the sample of steel placed in a furnace and heated. The method is not applicable industrially and a modified form has been developed. Burgess has described[1] the use of a neutral body and the differential couple, an arrangement first devised by Sir Roberts-Austen. The differential couple is mounted with one hot junction in the test piece and the other in a neutral body (one with no transformation points). The data obtained are curves of temperature of test piece *vs.* temperature difference between the test piece and the neutral body. These are mounted side by side in the furnace and heated and cooled as uniformly as possible.

The Brown transformation-point recorder is a roll-chart deflectional recorder in which the galvanometer moving element has two coils on one winding. One of these coils is connected to the couple in the test piece and the other is connected to the differential couple. In operation the recorder and furnace are started and the test-piece couple is permanently connected to the galvanometer. The differential couple is connected through an interrupting contact such that the record shows two rows of dots, one of which gives the temperature and the other the temperature difference measured by the distance apart of the two curves. Base-metal couples and rather large test pieces are used. The instrument will show a very marked jog at the eutectoid transformation point and by close observation the allotropic transformation point A_3 may be detected in pure iron.

The Leeds & Northrup transformation-point indicator is a semiautomatic recorder giving a continuous curve of temperature *vs.* temperature difference. The complete apparatus includes the recorder, two wall-type double-suspension galvanometers, furnace, rheostat and thermocouples. The recorder is the potentiometer type with a drum chart. The potentiometer slide wire is mounted on the same shaft as the

[1] G. K. Burgess, On Methods of Obtaining Cooling Curves. Bureau of Standards *Sci. Paper*, No. 99.

chart, which turns with it. A pen in a carriage can be moved across the chart by means of a long screw. One galvanometer is connected to the thermocouple in the potentiometer circuit and the other directly to the differential couple through an adjustable resistance. The drum is turned by hand to maintain a balance in the potentiometer circuit and with the other hand the pen is made to follow the deflections of the second galvanometer. This operation gives directly a temperature *vs.* temperature difference curve. The temperature coordinate is 20 in. long and the other is about 5 in. long. The periods of the galvanometers are sufficiently short to allow the heating and cooling to be done in less than 1 hr. The curve obtained is a continuous line which is slightly ragged due to manual operation, and is easily translated into metallurgical terms. It can be replotted into a curve of ϑ *vs.* $\frac{\Delta(\vartheta - \vartheta')}{\Delta \vartheta}$ the so-called "Derived Differential Curve" due to Rosenhain.[1]

Temperature Control.—The meaning of temperature control can be extended to cover not only the control of temperatures, but also the control of processes through a knowledge of the temperatures involved. In this sense it has a very wide interest. A list of the industries in which temperature control is used in one way or another would cover nearly the entire field of industry. This discussion is confined to the field of high temperatures.

Practically all pyrometric installations are designed as an aid in furnace or oven operation. Some of the factors which work difficulty in the regulation of furnaces, ovens, kilns, tanks, etc., are: (1) Inconstancy of heat supply. (2) Variation of internal absorption or generation of heat. (3) Variation of heat lost by radiation, convection, etc. (4) Unsteady supply or composition of material to be heat treated.

Each of these items is intimately connected with the temperature and temperature variations of the material being heat treated.

In the ordinary thermocouple installation, the couple is so mounted that the temperature of the furnace or the material within the furnace is obtained. For temperature control, however, it may be necessary to regulate the temperature at some different point, the temperature of which has a known or determinable significance and relation to the temperature at the point where the final control is required. For precise regulation the temperature should be directly controlled at a point where the effect of a variation in heat supply is registered with as little time lag as possible, and the temperature of this point may be so adjusted as to indirectly control the temperature of the material being heat treated. If a large time lag exists, a thermocouple may still indicate a low temperature for some time after the development of heat in the furnace walls is sufficient to produce a temperature much too high. In a similar manner the couple may read high for some time after the amount of heat in the furnace walls is insufficient. Hence, the temperature of the control couple oscillates about the correct temperature and the magnitude of the departure from the desired steady state can be reduced to a minimum only by making the time lag as small as possible.

Automatic Alarm.—A galvanometer of low resistance may be fitted with electric contacts made by the pointer which will ring a bell. No relay is necessary. The alarm should be used only when the departure from a certain temperature range results in real danger to life or property, or when a process is completed. Otherwise it becomes a nuisance.

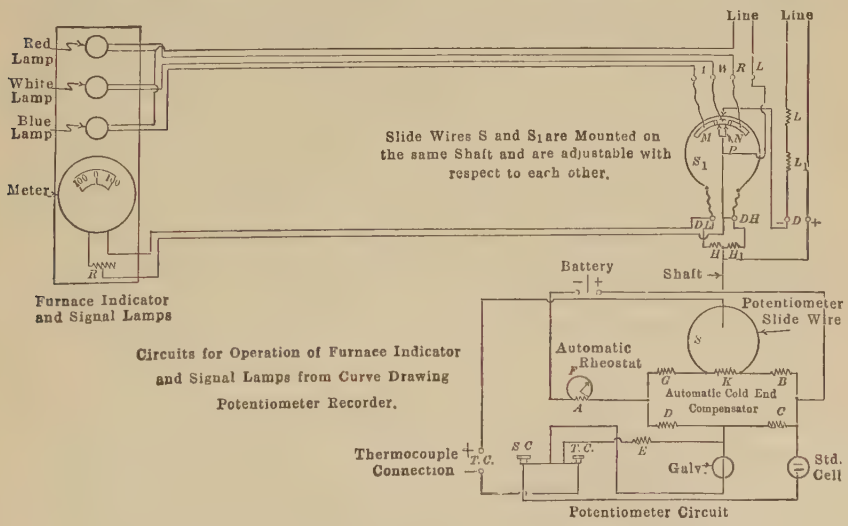
Manual Signaling.—This is of greatest use in extensive pyrometer installations having a central pyrometer station for indicators and recorders. Signals

[1] Observations on Recalescence Curves. *Phys. Soc. Lond.*, 21, p. 180, 1908.

with colored lights are sent from here to the furnace operators. The signals may be changed as frequently as necessary, and the measurements of temperature can be more reliable than when the indicators are placed in the furnace room.

Automatic Signaling.—A recorder may be easily modified to accomplish automatic signaling. With the galvanometric recorder the chopper bar instead of depressing the galvanometer pointer onto the chart makes an electric contact either through the insulated extremity of the pointer or by pressing together a double spring contact. These contact points are carried upon an arm which may be swung across the scale and adjusted for any temperature range. The independent electrical circuits containing the contact points are thus opened and closed depending upon the deflection of the galvanometer pointer, and may be employed for any mode of electrical signaling. Galvanometric recorders have not been very successfully constructed to record and signal at the same time, but the potentiometric recorder is easily made to do this. The latter instrument may be fitted with two or more contact points on its slide wire disc which touch a stationary contact when the slide wire is turned. Figure 32 shows diagrammatically the manner in which the movement of the slide wire serves to convey signals to the operator of a furnace. In addition to the colored signal lights the instrument may be equipped with a differential meter which indicates the departure of the furnace from the proper temperature. Such a mechanical arrangement for signaling does not interfere in any way with the mechanism for recording.

Automatic Temperature Control.—Indicators or recorders fitted with means for closing electric contacts are made by the Bristol Co., Brown Instrument Co., and the Leeds & Northrup Co. Automatic control requires the expenditure of



Circuits for Operation of Furnace Indicator and Signal Lamps from Curve Drawing Potentiometer Recorder.

FIG. 32.—Automatic signal system for temperature control.

more power than is required to operate signals, and relays are necessary between the controller and the furnace. Gas and oil valves are opened and closed by means of solenoid magnets, and electric switches are easily operated for electrically heated furnaces.

The Leeds & Northrup Co. has developed a two-couple continuous curve recorder for automatically heating furnaces to definite temperatures. As applied in carburizing and tempering of steel one thermocouple is placed in the furnace chamber and the other in contact with the steel piece. The controller automatically maintains the furnace for instance at 1,000°C. until the steel has reached 900°C. and then it drops the furnace to 900°C. Such an arrangement for control can be applied to a great many processes.

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SECTION XIII

REFRACTORIES

By A. V. Bleininger[1]

General Aspects.—The term "refractories" embraces all materials used in the arts for the construction of heat-resisting containers, using the word in its broadest sense, whether it be to afford space for the evolution of gases in combustion processes, or the holding of igneous liquids or of solids undergoing calcination. The two principal functions involved in the use of refractory materials are those of thermal insulation and conduction. In a furnace refractories serve the purpose of confining the heat and preventing an excessive loss to the atmosphere; in the case of a muffle or retort it is necessary that the heat be conducted through the walls to the charge as rapidly as possible.

Although, in general, the heat-resisting quality of refractories is of paramount importance, this is by no means the only requirement and may at times be only a secondary consideration. Refractories may be expected to resist high temperatures under negligible or again under heavy loads and stresses, to resist mechanical abrasion at various temperatures, to prevent the intrusion of molten metals, slags, glass, carbon or metallic vapors, to withstand sudden temperature changes and the action of superheated steam, hydrocarbons, sulphurous oxide, chlorine and other gases. Under one set of conditions high thermal conductivity may be required and under another high insulation value, while in still other cases good electrical resistance at moderately high temperatures may be demanded. It is manifestly impossible to expect that a single refractory will fulfill all of these functions with any degree of satisfaction and hence the proper selection of a suitable material becomes an important task.

The importance of the function of refractories in industry is not generally recognized and the subject has not received the attention to which its economic significance entitles it. This is at once apparent when one considers that heat-resisting materials are indispensable in the practice of the metallurgical arts, the generation of steam, the production of coke and gas, the melting of glass, the clinkering of cement, the firing of ceramic products and innumerable other industrial processes.

Since in many reactions the yield increases rapidly with increase in temperature above a given point it is obvious that a gain in working temperature of only a few degrees, made possible by the use of a more refractory material, may result in so much higher returns that any additional cost of the installation is negligible.

General Properties of Refractories.—It is generally understood by practical men that the virtues and faults of refractories for a given purpose are usually determined by one or two predominating qualities possessed by a material, other considerations being of secondary importance. Not infrequently a refractory, rated low from the heat-resisting standpoint, may be the most desirable material for some special use, say for gas producers, carburetors for blue gas, ladles for pouring steel, etc. It is desirable for this reason to consider the principal chemical and physical qualities called into play and to apply their consideration in the

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case of each type of refractory. At the same time we shall consider the methods employed for estimating these special properties since familiarity with them will enable us to differentiate the various materials and to recognize their limitations.

The principal factors involved in the use of refractories may be summarized as follows:

1. Chemical and mineralogical composition.
2. Refractoriness.
3. Permanent expansion or contraction upon reheating.
4. Porosity and specific gravity.
5. Resistance to compression at furnace temperatures.
6. Thermal properties, heat capacity, thermal conductivity and expansion.
7. Resistance to sudden temperature changes.
8. Resistance to slagging action.
9. Mechanical strength and resistance to abrasion.
10. Electrical resistance and dielectric strength.

Chemical Composition.—The classification of refractories for metallurgical purposes has been principally based on the composition of the materials as expressed by their acidity, *i.e.*, the ratio of silica to the basic constituents. It was assumed that siliceous slags would not attack refractories high in silica, and basic slags would not combine with basic refractories. In a general way this assumption is undoubtedly true, but it must be realized that such properties as density are of great importance in this respect. Porous refractories, no matter how well adapted they may be from the chemical standpoint to resist certain slags, are poorly suited for such requirements since they absorb any liquid by capillary action and hence are destroyed in a short time. On the other hand, in the calcination of solids, an excellent showing may be made by refractories which represent, chemically, exactly the opposite characteristics of the charge. Thus in the burning of lime, silica brick or quartzite kiln linings have given very excellent results. This obviously is due to the small reacting surfaces offered by the limestone and the comparatively low temperature involved.

At high temperatures the reaction between solids is much more active and diffusion phenomena more rapid. In the open-hearth furnace, therefore, it is necessary to separate the acid silica brick from the basic magnesite by means of a neutral parting of chromite.

It is evident that chemical composition is of fundamental importance in determining the character and quality of a refractory. A low content of basic impurities in siliceous materials and of acid ones in basic refractories is obviously desirable from the standpoint of heat resistance and for this purpose chemical analysis is of great importance in determining the causes of success or failure. In refractories used at temperatures close to their softening point the presence of impurities amounting only to 1 or 2 per cent may cause failure. The chemical analysis of refractory materials requires careful work and the modern analytical methods proposed by Hillebrand, the Geophysical Laboratory of the Carnegie Institution, Mellor, and others should be employed.

The range of materials used as refractories is very wide and includes such substances as the fireclays and kaolins, quartzites, bauxites, schist, magnesite, dolomite, graphite, carborundum, fused alumina, chromite, magnesium aluminate, magnesium silicate, zirconium oxide, zirconium silicate, boron nitride, and others.

The results of the chemical analysis are sometimes computed in the form of empirical formulas which simplify comparison of the materials with recognized standards

and bring out the fundamental chemical characteristics. In the case of aluminum silicates the aluminum oxide is usually made equal to unity in the formula and the fluxing oxides combined in one expression so that the entire chemical composition is represented by only two variables, the RO and R_2O oxides, and the silica. With basic refractories the principal constituent is made equal to unity. Thus for magnesite the formula would be: $xR_2O \cdot RO \cdot yR_2O_3 \cdot zSiO_2$, where the R_2O is usually negligible.

The calculation involved consists in dividing the per cent of each constituent by its molecular weight, adding the quotients or molecular equivalents of each group and dividing every quotient by the equivalent of the oxide or group of oxides to be taken as unity. In the case of fire clays the alumina equivalent is taken as unity. The result is then written in the form of the expression: $xRO \cdot R_2O_3 \cdot yRO_2$. The iron oxide is usually given in the ferrous form. This calculation is illustrated by the following example, dealing with the chemical composition of a New Jersey firebrick, the first column of figures showing the analysis, the last the quotient obtained by dividing the per cent of each compound by its molecular weight in round numbers:

Silica	77.82 per cent	÷ 60	= 1.296	
Alumina	19.00 per cent	÷ 102	= 0.186	
Ferric oxide	1.01 per cent	÷ 160	= 0.00625 × 2 = 0.0125	(ferrous oxide)
Titanium oxide	1.65 per cent	÷ 80	= 0.0206	
Lime	0.22 per cent	÷ 56	= 0.0039	
Magnesia	0.06 per cent	÷ 40	= 0.0015	
Soda	0.10 per cent	÷ 62	= 0.0016	
Potash	0.28 per cent	÷ 94	= 0.0030	

Dividing through by the equivalent of alumina, 0.186, we have:

Silica	1.296	÷ 0.186	= 6.96700
Alumina	0.186	÷ 0.186	= 1.00000
Ferrous oxide	0.0125	÷ 0.186	= 0.06720
Titanium oxide	0.0206	÷ 0.186	= 0.11070
Lime	0.0039	÷ 0.186	= 0.02096
Magnesia	0.0015	÷ 0.186	= 0.00806
Soda	0.0016	÷ 0.186	= 0.00806
Potash	0.00298	÷ 0.186	= 0.01602

Adding the values obtained for the five fluxing constituents, we have:

Ferrous oxide (FeO)	0.06720
Lime	0.02096
Magnesia	0.00806
Soda	0.00806
Potash	0.01602
	0.12030

Arranging the three groups in the empirical formula, we obtain:

$$\left.\begin{matrix} 0.016\ K_2O \\ 0.008\ Na_2O \\ 0.067\ FeO \\ 0.021\ CaO \\ 0.008\ MgO \end{matrix}\right\} \quad 1Al_2O_3 \quad \left\{\begin{matrix} 6.97\ SiO_2 \\ \\ 0.11\ TiO_2 \\ \\ \end{matrix}\right.$$

Reducing this expression to its simplest terms we obtain:

$$0.12\ RO \cdot Al_2O_3 \cdot 6.97\ SiO_2$$

It is evident that we are dealing here with a siliceous clay having an excess of 4.97 molecular equivalents of free silica over and above the combined silica of the

clay substance, $Al_2O_3 \cdot 2SiO_2 \cdot 2H_2O$. The clay therefore consists essentially of 258 (corresponding to molecular weight of the clay substance) parts, by weight, of clay substance and $4.97 \times 60 = 298.2$ of free silica or sand.

The mineralogical structure of the heat-resisting materials is of great importance in the study of this subject and the application of the petrographic microscope both to the examination of thin sections and of fine grains is gaining in recognition more and more. It is impossible without this aid to aquire a thorough knowledge of the conversion of quartz to cristobalite and tridymite in silica brick, the formation of sillimanite in clay firebrick, the crystallization of spinels, the recrystallization of periclase in magnesite brick, the formation of accessory minerals, etc. Both the chemical and mineralogical composition of refractories will be considered in dealing with the several types of refractories.

Refractoriness.—It is evident that a refractory retains its usefulness only as long as it continues to be sufficiently rigid to support its own weight and such loads as may be imposed upon it in furnaces.

Unfortunately, the term "refractoriness" has no specific meaning. Generally, it is supposed to represent the so-called melting point of a material. This confronts us with a serious difficulty due to the fact that the substances used for work of this kind possess no melting point in its proper physical conception. It is a well-known fact that silicates, like clays, lacking in well-defined crystalline structure and of high molecular viscosity, offer no definite transition point from the solid to the liquid phase, nor any other criterion, corresponding to the transformation of a substance from the anisotropic to the isotropic state. We are compelled therefore to depend upon deformation data, such as the rounding of edges, the bending of specimens of standard size and shape in the manner of Seger cones, or the formation of drops. But even when a standard condition of deformation has been established it is evident from theoretical considerations that the rate of heating will have a decided influence upon the deformation temperature, rapid firing resulting in a higher fusion point than that with a slow rate of heating. This condition is made still more complicated by the heterogeneous nature of the materials to be tested, which consist of particles of different substances varying greatly in size. It is thus seen that the determination of the softening temperature is not so simple as it might appear. Fortunately, the higher the temperature involved, the less marked seems to be the influence of the rate of heating upon the deformation point.

In making a softening point determination of a refractory it seems desirable to grind the specimen so that the material will pass the 80-mesh sieve and to mold from it, by the use of dextrin or other organic glue, small tetrahedra, $\frac{3}{4}$ in. high and $\frac{1}{4}$ in. wide at the base.

It is hardly necessary to point out that any metallic iron introduced by the grinding should be removed by means of a magnet. The tetrahedral specimens are placed on a plaque, made from a mixture of kaolin and sintered or fused alumina in the proportion of 1 : 1 in the case of clay and silica refractories. For basic specimens a mixture of sintered and low-fired magnesite, or of pulverized chromite, may be employed. Occasionally crushed carbon is used cemented together with tar or other organic cementing substances. The test cones are placed in position, alternating with the standard pyrometric cones (manufactured by Prof. Edward Orton, Columbus, Ohio). Usually three or four numbers are employed, representing as many softening points.

The object of the test then is to raise the temperature of the furnace to a point at which the specimen deforms by bending until its apex touches the plaque or in the case of certain refractories shows distinct evidence of fusion. The softening point of one of the standard cones indicated by the prescribed degree of bending establishes

the so-called melting point of the refractory. One of the essential requirements of the test is that the rate of heating does not exceed 5°C. per minute, controlled preferably by the use of an optical pyrometer of the Morse or Holborn-Kurlbaum type.[1]

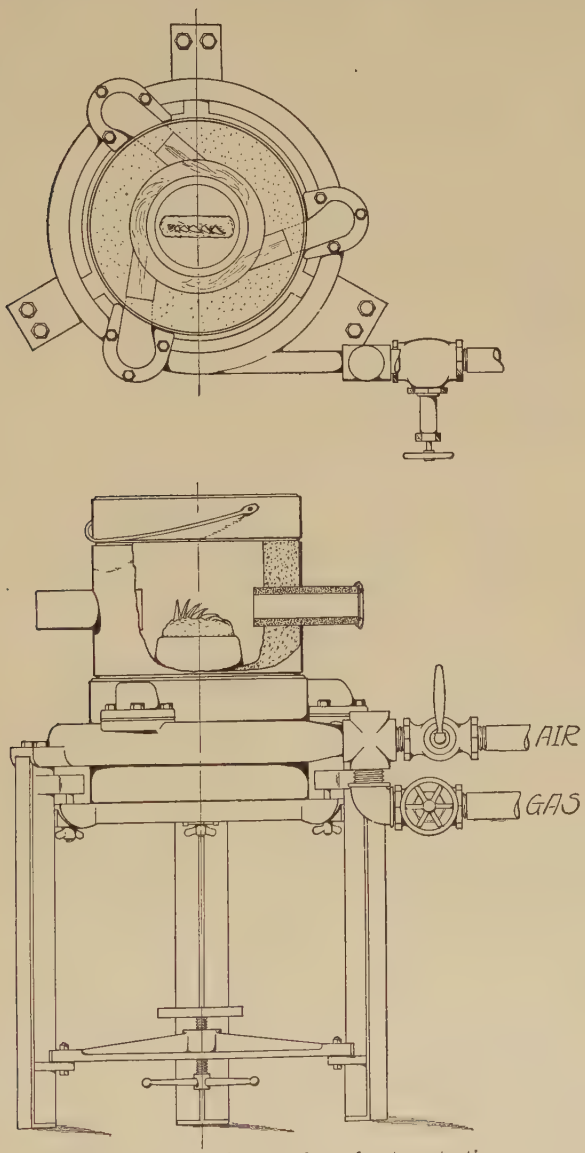

FIG. 1.—Gas furnace for refractory testing.

In gas-fired furnaces there is not much danger of heating the specimens too rapidly since the rise in temperature when approaching 3,000°F. is slow, but in electric furnaces this is a point which must be watched.

[1] See p. 449.

The furnaces commonly used are simple in construction and fired with gas wherever possible, but for testing magnesite and other refractories it is necessary to make use of electrical furnaces. Clay and silica refractories are most conveniently tested in the type of furnace illustrated in Fig. 1 It consists of an iron casing lined with a high-grade refractory and a pedestal upon which the test cones are placed. The lining and containers are best made from a mixture of fused alumina and kaolin in the proportion of 2 : 1. In order that the required temperatures may be reached it is necessary to use compressed air with the gas, employing a Fletcher burner. The air pressure need not exceed 20 lb. per square inch. Preheating the air by passing it through a section of 4-in. pipe kept hot by means of an auxiliary burner assists greatly in reaching the desired point. Where natural or city gas is not available vaporized gasoline can be used. In case it is necessary to resort to the use of petroleum the furnace must be made considerably larger to provide space for the combustion of the oil vapor.

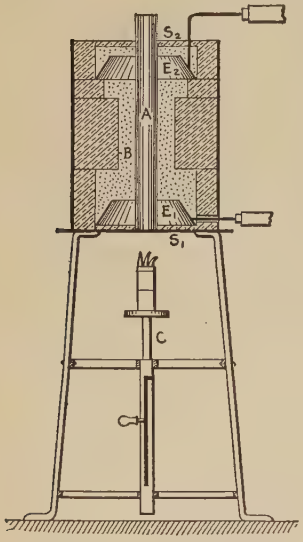

FIG. 2.—Electrical furnace.

The electric furnaces to be used for temperatures much above the melting point of platinum may consist of a tube made from fused magnesia wound with molybdenum or tungsten wire. Such furnaces require that an atmosphere of hydrogen be maintained within to prevent the oxidation of the heating element. For still higher temperatures the Arsem furnace may be employed, consisting essentially of a graphite spiral as the heating element within which a tube of fused magnesia serves as the container of the specimens to be tested. The furnace is enclosed in a gastight jacket and a vacuum is maintained by means of a pump. The specimen is observed through a glass looking down into the tube and the softening temperature determined by means of the Holborn-Kurlbaum optical pyrometer. Usually there is no room for placing the control cones.

Electrical furnaces of the types just mentioned require considerably more attention and care than they can receive in works laboratories and it is advisable to employ apparatus of a simpler type. A furnace of this kind is shown in Fig. 2, consisting essentially of a tube, A, pressed from granulated, fused magnesia mixed with linseed oil. Somewhat impure magnesia is to be preferred, containing from 5 to 10 per cent of ferric oxide, silica and alumina. Around this tube open at both ends, in the space B, granular electrode carbon, varying in size between grains passing the 12- and 20-mesh sieves, is placed, the cross-section being constricted at the middle of the tube. This increases the resistance throughout the section and causes the temperature to be correspondingly higher. The electrodes E_1 and E_2 are rings of wrought iron connecting to the leads. The cones are placed upon a refractory cylinder which is fastened to a disc arranged to move up or down. The carbon is confined by rings of refractory material made preferably from a mixture of alundum and kaolin. There are on the market also well-constructed furnaces such as the Hoskins and Northrup types which are adapted for this work.

It is common practice to state the softening point of a material by saying that it corresponds to that of a certain cone number, or we may say that a clay softens between, say, cones No. 31 and 32. The practice of referring to the published softening temperatures rather than the cone numbers is not to be recommended, especially as the values usually given are derived from old sources and have been shown to be

incorrect. Temperatures should be given only when the softening point has been determined direct by means of the optical pyrometer.

The composition and estimated softening temperatures of the American cones usually employed in the testing of refractories are given in Table 1. It is necessary to emphasize the fact that the softening temperatures are by no means fixed but vary according to the rate of heating, being lower under conditions of slower firing.

TABLE 1

Number	Formula as given by Seger, Hecht and Cramer	Softening temperature as given by makers		From corrected data by Kanolt, degrees Centigrade
		°C	°F	
1	0.3 K_2O, 0.7 CaO, 0.2 Fe_2O_3, 0.3 Al_2O_3 4 SiO_2	1,150	2,102	
2	0.3 K_2O, 0.7 CaO, 0.1 Fe_2O_3, 0.4 Al_2O_3 4 SiO_2	1,170	2,138	
3	0.3 K_2O, 0.7 CaO, 0.05 Fe_2O_3, 0.45 Al_2O_3 4 SiO_2	1,190	2,174	
4	0.3 K_2O, 0.7 CaO, 0.5 Al_2O_3 4 SiO_2	1,210	2,210	
5	0.3 K_2O, 0.7 CaO, 0.5 Al_2O_3, 5 SiO_2	1,230	2,246	
6	0.3 K_2O, 0.7 CaO, 0.6 Al_2O_3, 6 SiO_2	1,250	2,282	
7	0.3 K_2O, 0.7 CaO, 0.7 Al_2O_3, 7 SiO_2	1,270	2,318	
8	0.3 K_2O, 0.7 CaO, 0.8 Al_2O_3, 8 SiO_2	1,290	2,354	
9	0.3 K_2O, 0.7 CaO, 0.9 Al_2O_3, 9 SiO_2	1,310	2,390	
10	0.3 K_2O, 0.7 CaO, 1.0 Al_2O_3, 10 SiO_2	1,330	2,426	
11	0.3 K_2O, 0.7 CaO, 1.2 Al_2O_3, 12 SiO_2	1,350	2,462	
12	0.3 K_2O, 0.7 CaO, 1.4 Al_2O_3, 14 SiO_2	1,370	2,498	
13	0.3 K_2O, 0.7 CaO, 1.6 Al_2O_3, 16 SiO_2	1,390	2,534	
14	0.3 K_2O, 0.7 CaO, 1.8 Al_2O_3, 18 SiO_2	1,410	2,570	
15	0.3 K_2O, 0.7 CaO, 2.1 Al_2O_3, 21 SiO_2	1,430	2,606	
16	0.3 K_2O, 0.7 CaO, 2.4 Al_2O_3, 24 SiO_2	1,450	2,642	
17	0.3 K_2O, 0.7 CaO, 2.7 Al_2O_3, 27 SiO_2	1,470	2,678	
18	0.3 K_2O, 0.7 CaO, 3.1 Al_2O_3, 31 SiO_2	1,490	2,714	
19	0.3 K_2O, 0.7 CaO, 3. Al_2O_3, 35 SiO_2	1,510	2,750	
20	0.3 K_2O, 0.7 CaO, 3.9 Al_2O_3, 39 SiO_2	1,530	2,786	
23	0.3 K_2O, 0.7 CaO, 5.4 Al_2O_3, 54 SiO_2	1,590	2,894	
26	0.3 K_2O, 0.7 CaO, 7.2 Al_2O_3, 72 SiO_2	1,650	3,002	1,600
27	0.3 K_2O, 0.7 CaO, 20.0 Al_2O_3, 200 SiO_2	1,670	3,038	1,620
28	Al_2O_3 10 SiO_2	1,690	3,074	1,635
29	Al_2O_3 8 SiO_2	1,710	3,110	1,650
30	Al_2O_3 6 SiO_2	1,730	3,146	1,670
31	Al_2O_3 5 SiO_2	1,750	3,182	1,685
32	Al_2O_3 4 SiO_2	1,770	3,218	1,705
33	Al_2O_3 3 SiO_2	1,790	3,254	1,720
34	Al_2O_3 2.5 SiO_2	1,810	3,290	1,740
35	Al_2O_3 2.0 SiO_2	1,830	3,326	1,755
36	Al_2O_3 1.5 SiO_2	1,850	3,362	
37		1,880	3,416	1,835
38		1,960	3,470	1,850
39		1,950	3,542	1,880
40		1,980	3,596	1,915
41		2,010	3,650	1,970
42	Al_2O_3	2,050	3,722	2,025

The use of Seger cones for the estimation of the softening point of basic refractories is not advisable and the optical pyrometer should be employed for this purpose, especially since the temperatures involved are often above the available cone series. When it is necessary to resort to the use of cones these should be placed on a plaque made from a mixture of fused alumina and kaolin and not upon that supporting the test specimens which is necessarily basic.

The Holborn-Kurlbaum pyrometer must be carefully calibrated for the particular absorption glass in use. It should be kept in mind that the lamp needs to be recalibrated from time to time. The calibration methods of Kanolt are to be recommended for this purpose.[1]

In work involving direct pyrometric determinations it is usually not feasible to take the bending of the test cones as the softening point but the end point is considered to have been reached when a distinct rounding of the edges is observed or a bead is formed. At best the estimation of the so-called fusion or softening point is not so easy a matter as might appear, both owing to the difficulty of close observation and the fact that at very high temperatures volatilization becomes quite marked, manifesting itself in the apparent subliming or melting away of the test specimen, especially with magnesia or its compounds. The effect of the intense reduction almost unavoidable in the electric furnace also is very prominent so that the real melting point of the substance under test can not be found, due to the reduction of the iron oxide commonly present to metallic iron as well as its actual volatilization. Silica also is driven off in appreciable amounts and any alkalis present are lost completely.

Where the character of the refractory permits it the fusion test should be made in the gas-fired furnace under neutral or at least not excessively reducing conditions. The electric furnace then remains for the testing of materials fusing above the melting point of platinum, 1,755°C.

Permanent Expansion and Contraction.—Practically all refractories are subject to further changes in volume when heated for longer periods under the conditions of their use. These volume changes are called permanent in distinction from those brought about by thermal expansion and contraction. Clay, magnesite and alumina refractories tend continually to contract or shrink while the siliceous materials expand more or less. It is evident that excessive volume changes are exceedingly undesirable inasmuch as they tend to bring about a breaking down of the furnace structure, through the opening up of joints, and the formation of cracks, thus causing leakage and ultimate collapse of the furnace. For most purposes materials showing as small a contraction or expansion in use are most desirable. Excessive shrinkage may be due to insufficient firing or to lack of refractoriness. For most purposes the harder a refractory is fired in its manufacture the more satisfactory it is.

The contraction of a material with temperature may be employed as a means of estimating its refractoriness as well as for determining the proper point to which it should be fired. This affords a method of study which is exceedingly valuable and in fact makes possible a classification of refractories and the discrimination between the various grades. In securing the data needed for this purpose it is only necessary to prepare a series of specimens from the unburned material of such size as will permit accurate measurement of volume and to fire them at a number of increasing temperatures. We might, for instance, mold and dry 12 briquettes, 4 by 3 by 3 cm., immersing them in petroleum until completely saturated and then determine their volume by any convenient method. After heating the specimens to a low temperature just sufficient to expel the petroleum they are placed in a suitable test kiln or furnace capable of good temperature control and provided with calibrated platinum-platinum rhodium thermocouples and a high-resistance millivoltmeter or, still better, a potentiometer. The temperature range must be adjusted to the kind of refractory being tested. If we are dealing with a fireclay material the range between 1,150° and 1,425°C. may be chosen. The heat is raised from above 1,000°C. at the rate of 30°C.

[1] Bureau of Standards, *Technologic Paper* No. 10, see also p. 450, this book.

per hour and this schedule must be adhered to closely in order that it may be possible to check the results in other firings. The specimens must be so placed that they can be reached from without the furnace and withdrawn by means of suitable tongs. It is also well to place a series of pyrometric cones as close to the briquettes as possible to serve as a check on the heat treatment. When the pyrometer indicates the temperature of 1,150°C. the first briquette should be withdrawn and at once placed in an auxiliary muffle furnace maintained at a temperature of about 600 to 700°C. If the temperature interval decided upon is 25°C. the next specimen must be removed at 1,175° and so on until the maximum temperature of 1,425° has been reached. The temperature intervals may be taken as large or small as is convenient. Where but few points are desired they may be made as large as 100°; a convenient interval is 40° which roughly approximates two standard cones. When all of the briquettes have been removed and placed in the auxiliary muffle the latter is allowed to cool down at its natural rate. When cold the specimens are marked according to the temperature at which they have been withdrawn and their volumes determined.

For the purpose of securing comparable records it is customary to compute the percentage of shrinkage of each specimen based upon its volume in the dried state

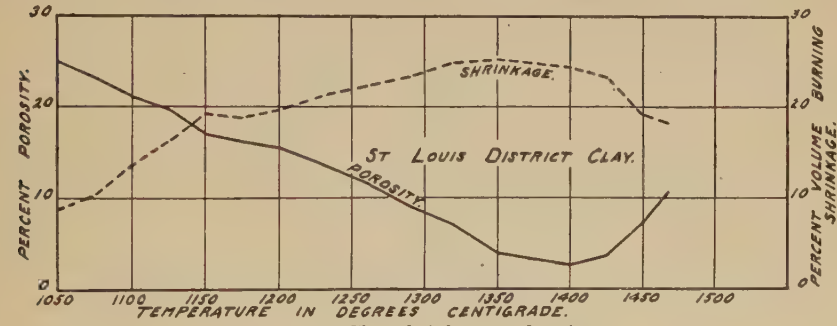

FIG. 3.—Clay shrinkage on heating.

according to the obvious relation: $100(v_1 - v_2)/v_1$, where v_1 = volume of briquette in the dried state and v_2 = its volume in the fired state. The plotting of these results in graphical form offers a convenient form of comparison from which an estimate of the behavior of the material can be readily made. Such a graph is shown in the upper curve of Fig. 3 which at once tells us that the refractory continues to contract until the temperature 1,350°C. has been reached. The material therefore must be burned to this point if it is desired that it be as constant in volume as possible. It will be noted that beyond this temperature the shrinkage decreases which is due to expansion of the mass, and the downward trend of the curve is equivalent to negative shrinkage, or expansion. This phenomenon is very characteristic of clays which "overfire," *i.e.* which on account of excessive softening due to heat and the evolution of gases develop a vesicular or spongy structure accompanied by a swelling of the mass. Clay in this state suffers a decided deterioration in quality since it flows and deforms, possesses low mechanical strength and offers but little resistance to corroding influences such as slags. The diagram thus gives important data concerning a refractory, namely the rate of shrinkage, with temperature, dv/dt, the temperature at which maximum condensation has taken place and the point at which "overfiring" begins.

It is evident that such studies show also volume alterations due to molecular changes, such as the transformation of quartz to cristobalite and tridymite, the decomposition of clay into sillimanite, etc.

Equipment for Volume Studies.—The volume of the specimens may be determined by means of several methods of which one or the other may be the more convenient. The direct determination of volume is made most conveniently by means of a voluminometer of the Seger type, illustrated in Fig. 4. The apparatus is first filled with kerosene to the zero mark of the burette which coincides with the zero mark of the tube, attached to the cap of the bottle.

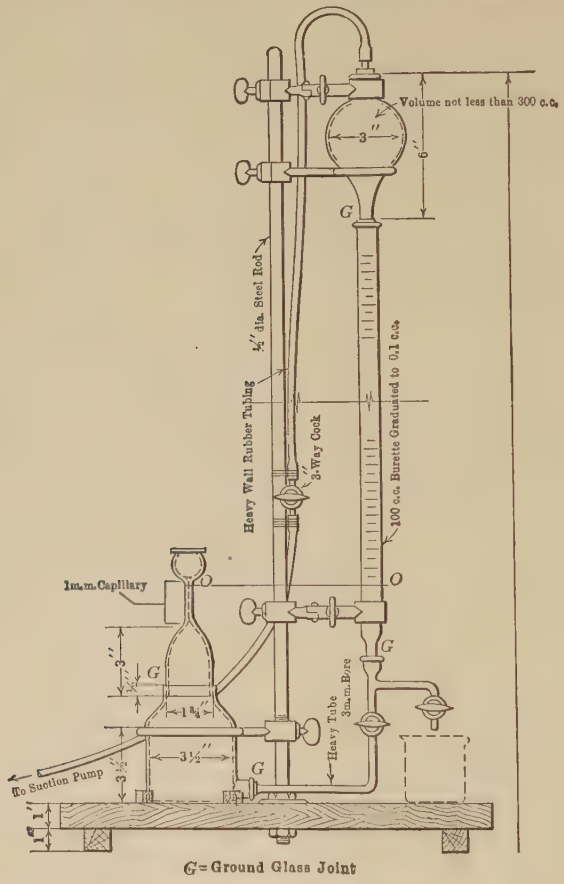

FIG. 4.—Schurecht voluminometer.

Sufficient of the liquid to allow for the introduction of the specimen is then drawn up into the burette and bulb on top of the latter and the stop cock placed in the shutoff position. The cap is removed, the specimen previously saturated with petroleum placed in the bottle, the cap replaced and the liquid allowed to run back into the bottle up to the zero mark. The stop cock is closed again and the volume of the test piece read from the burette which is graduated to 0.1 c.c.

Indirectly, volume may be determined by two weighing methods. One method consists in saturating the specimen by immersion in a suitable liquid, kerosene or water, and weighing it in this state. The same test piece is then weighed while suspended in the liquid. From these weights the exterior volume of the specimen, in cubic centimeters, is computed from the relation: $v = (w - s)/\delta$, where v = exterior

volume, w = saturated weight, s = suspended weight, in grams, and δ = density of the liquid.

A modified pycnometer has also been used by Schurecht[1] for the determination of the volume of test specimens. It consists essentially of a wide-mouth cylindrical glass vessel provided with a ground in, hollow stopper which has a tube in its center. The volume of this piece is computed from the relation: $v = (w - w_1 + b)/\delta$, where v = volume of specimen; w = weight of bottle plus liquid in grams; w_1 = weight of bottle plus liquid plus specimen; b = weight of saturated test piece, and δ = specific gravity of liquid.

With increase in temperature above a minimum point the apparent volume of most refractories decreases steadily, due to colloidal changes or to the influence of fluxes which tend to bring about softening of the mass. In general, most refractory oxides undergo contraction upon being heated which in a large measure must be described as a surface-tension phenomenon. This force is active in reducing the superficial area of the mass and theoretically tends to reduce the volume to the shape of a sphere. To use a homely simile the mass contracts as if it were uniformly acted upon by a tension similar to that of rubber stretched upon a ball. At lower temperatures the rigidity of the system is too great to cause appreciable contraction but at higher heats the effect becomes more and more manifest as may be readily demonstrated by heating alumina, magnesium oxide, thorium oxide, zirconium oxide, etc., to temperatures greatly below their fusion points proper. This contraction increases rapidly in the presence of fluxes so that for instance the addition of a small amount of boric acid will cause alumina to reach constant volume at 1,750°C., about 300° below its softening or melting temperature. The presence of basic oxides in clay and siliceous materials and of acid ones in the basic refractories tends to form eutectic combinations which depress the softening temperature of the whole and at the same time bring about a marked decrease in the viscosity of the system. This condition may cause marked shrinkage at furnace working temperatures. For this reason it is desirable that the refractories be fired, if possible, to a temperature higher than that at which they are used.

Porosity.—Coincident with the shrinkage in volume there occurs contraction of the pore space. The reduction in porosity may therefore be taken as a criterion of the gradual softening of the mass and being determined quite readily it is often used for the purpose of establishing the refractory character of a material. This method is not so sensitive with reference to small temperature effects as the measurement of volume but is amply accurate for most purposes. The determination of porosity involves the weighing of the fired specimen in air, weighing the piece when completely saturated with kerosene or water and finally when suspended from the beam of the balance and immersed in the liquid at a definite temperature. The first determination requires that the specimen be previously dried to remove any moisture. In bringing about complete saturation the piece is usually allowed to boil in water under a partial vacuum for 5 hr. and is then allowed to cool in water down to the normal temperature. The porosity, or pore space, expressed in per cent of the volume of the specimen, is computed from the Purdy formula:

Per cent $p = 100(w - d)/w - s$, where,

w = weight in air of specimen, saturated with water, in grams,

d = weight in air of dry specimen,

s = weight of specimen suspended in water, and

per cent p = pore space, in per cent of the external volume.

[1] *Journ. Ceram. Soc.*, Vol. 1, p. 556–58.

Similar to the change in volume the porosities of a material fired to different temperatures may be plotted against the temperature. The curves thus obtained are equally instructive and valuable, for the comparison of the pyro-technical properties of refractories. The lower curve of Fig. 3 gives the results obtained with a fireclay material. Here again the rate of porosity decrease, the temperature at which the structure has become dense, shown by the approach to zero porosity, and the evidence or overfiring offer data of practical importance. The accuracy of the results obtained depends of course upon the accuracy of the temperature measurements. Too much attention cannot be given to the calibration of the pyrometers.

Since the determination of the porosity depends to a large extent upon the absorption of water by the material it will suffice in many cases to make use only of this value in establishing the relation between temperature and its effect upon the consolidation or vitrification of the body. In determining the absorption we need to know only the weights of the dry specimen and its weight when saturated with water, and the computation consists of the evident relation $100(w - d)/d$, where w = weight of saturated specimen, and d = weight of dry test piece. The value merely expresses the weight of water absorbed in terms of the dry weight of the specimen. The porosity value is to be preferred in all cases where closer differentiation is desirable or where the comparison of bodies possessing different specific gravities is involved.

Occasionally it may be desirable to determine the total porosity of refractories, especially where they are dense and the absorption of water is incomplete or where enclosed pores exist into which the liquid cannot penetrate. For the calculation of this value it is necessary to determine the bulk and the true specific gravity. The former is determined by obtaining the weight of the dried piece, in air, and its exterior volume, as described in a preceding paragraph. The bulk density obviously is equivalent to the relation w/v, where w = weight of specimen and v = its volume. The true specific gravity is found by crushing the material, passing it through the 100-mesh sieve and determining the values sought by means of the pycnometer in the usual manner. The true porosity is then computed from the relation: $p = 100(1 - d_1)/d_2$, where p = per cent pore space in terms of the exterior volume, d = true specific gravity and d_2 = bulk specific gravity.

Specific Gravity.—It is interesting to note that most silicates upon being heated increase in specific volume, that is, decrease in density, in spite of the fact that the exterior volume of the piece may contract. It is evident that this phenomenon is to be ascribed to molecular changes and to the progress of softening or fusion. In general, the porosity-temperature curves parallel the specific gravity-temperature plots. Fireclays higher in fluxing impurities invariably show a lower specific gravity at the same temperature than purer materials, but even refractory clays exhibit an appreciable drop in density. Thus a high-grade Kentucky flint clay of specific gravity 2.60, when fired to 1,550°C., possessed a density of 2.38. From the technical standpoint the determination of the specific gravity is often of service in quickly estimating the degree of firing to which a refractory has been subjected for the purpose of judging whether it has been brought to the proper end point, especially in the case of silica and magnesite brick.

The true specific gravity is also of great value in theoretical studies for the observation of transformation points. Since fusion from the exact standpoint represents a change of state which may be located by the discontinuity of some physical property, it may represent the intersection of the vapor-tension curves of the crystalline and

amorphous states, the point of optical deorientation, the break in electrical conductivity, the discontinuity of the heating curve, the change in specific volume, etc. For this reason plots showing the relation between specific gravity and temperature frequently establish critical points of considerable interest. At the same time it might be understood that even the most refined methods of the laboratory will fail in dealing with systems of great viscosity in which the high internal friction may arrest or greatly delay any changes by means of which the critical point is recognized.

Resistance to Compression at Furnace Temperatures.—In use refractories are frequently subjected to compression and sometimes tension stresses. This is particularly the case in the arches and crowns of furnaces and kilns, gas-retort benches, hot-blast stoves, etc. Any decrease in rigidity suffered by the refractory body due to incipient softening will at once become evident by more or less marked deformation under stresses exceedingly small compared with the compressive strengths of the material in the cold state. Thus, a firebrick showing a crushing strength in the cold of 3,000 lb. per square inch may be deformed under a pressure of 50 lb. per square inch at furnace temperatures. The softening of such substances through a given viscosity range is not only a function of temperature but of time as well. It is possible therefore to vitrify a body by exposure to higher temperatures in a shorter time and at lower points of the temperature scale during longer periods.

Great differences exist as to the viscosity and deformation of different minerals and rocks both natural and artificial. As a general proposition it may be stated that the higher the material is in fluxing impurities, *i.e.* basic oxides in the case of siliceous and acid oxides in basic mixtures, the greater the deformation under conditions of pressure. It is a fact also that silicates high in alumina show greater contraction under constant pressure than siliceous ones. The resistance of refractories to pressure at furnace temperatures therefore offers a measure of the rigidity of refractories and, indeed, of their resistance to heat, since it is evident that a material showing but little deformation at a given temperature, say 1,350°C., and under a given load is to be preferred to one which softens and gives evidence of excessive contraction.

With reference to resistance to load conditions it may be said that the refractoriness of a material is a function of the pressure employed. For instance, a firebrick, showing in the unloaded condition a softening point corresponding to cone 32 to 33, may under pressure show a steady deformation or collapse at a temperature represented by cone 19 to 20.[1] The initial mechanical strength and the degree of firing are, however, factors in determining the resistance to load conditions, since dense brick, formed under a heavy pressure, are more resistant than those of a more open and porous structure. The amount of cementing material present and the degree in which it coheres with the granular constituents is involved as well. Almost invariably hard-fired bricks in which this condition is brought to a high degree of perfection will stand up much better than those burned to a lower temperature. It will be seen therefore that resistance to pressure at furnace temperature is by no means a simple property and a test involving such a condition should not be prescribed indiscriminately. It should be reserved for refractories which are actually to be used in places where pressure is really an important factor. The test as applied so far in this country errs on the safe side but is liable to be unfair to certain materials. This defect is diminished by reducing the load applied and for this reason it is becoming customary to employ a pressure of 25 lb. per square inch rather than 50 lb. as has been the former practice.

Failure may occur in one of two ways, either through the softening of the mass as evidenced by excessive contraction, distortion or bending or through shear, in which

[1] J. W. Mellor and W. Emery, *Trans.* English Ceramic Soc., Vol. **17**, part 2, pp. 360 *et seq.*

case the failure is usually more or less abrupt. The former type of failure is more common in fireclay, the latter with silica, chromite and magnesite refractories.

The test as usually carried out is very simple and consists in applying pressure upon a standard size firebrick, placed on end within a suitable furnace, and raising the temperature at the prescribed rate. The maximum temperature is maintained con-

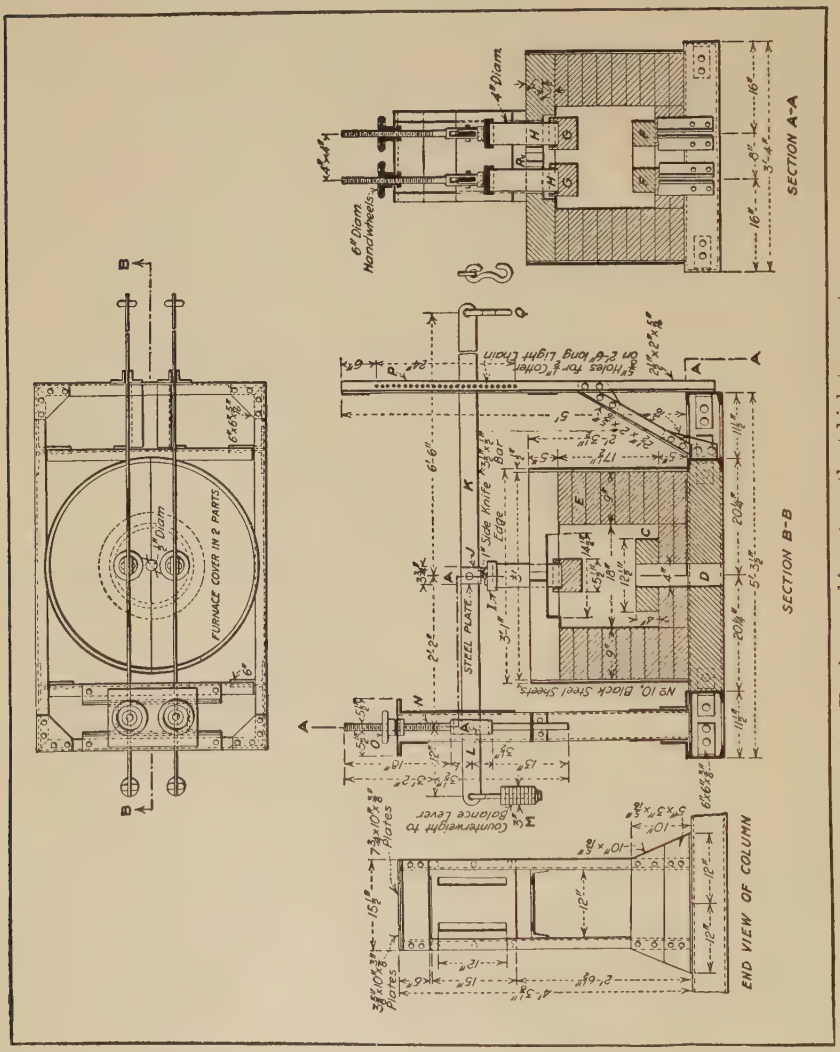

FIG. 5.—Testing crushing strength at high temperatrues.

stant for some time, usually 1½ hr. The furnace recommended for this purpose is shown in Fig. 5, in which provision is made for the simultaneous testing of two specimens. The expansion and contraction may be followed by attaching a pantograph recorder to the beam but the specimen is rated according to the contraction it has uffered expressed in per cent of the original length or whether it has failed by collapse.

The maximum temperature to be employed depends upon the kind of refactory to be tested. It is usually taken to be 1,350°C. for grade one firebrick, 1,400°C. for silica

and 1,450°C. for magnesite brick. However, no definite standards have been adopted by any official organization for general use.

A modified pressure test has been proposed by Nesbitt and Bell in which the specimens are heated to 1,350°C. and immediately subjected to pressure transmitted through a 2-in. steel ball. The depth to which the ball has been pressed into the specimen is considered to be the criterion of the load resistance of the material.

The results of tension tests upon refractories in the hot state are not available in the literature nor are data relating to the transverse strength of bricks and tiles. This is to be regretted since in many furnace constructions transverse loads must be considered. Again, it seems very probable that the compression test of firebricks will ultimately be replaced by one involving transverse stress.

Thermal Properties.—The thermal qualities of refractories, specific heat, conductivity and expansion are determined according to the established physical methods. It is evident that these properties are of considerable practical importance. The data available, however, on these subjects are quite meager, especially if it is considered that the structure of the manufactured product, irrespective of its chemical nature, is of paramount importance. Furthermore, these properties are subject to change with temperature and comparatively few constants are at hand to illustrate the character of these relations. It is known that the specific heat and thermal conductivity increase with temperature but the fundamental laws governing these changes have not been established. Furthermore, it must be realized that the structure of all these materials is certain to undergo physical changes which affect the thermal qualities.

Specific Heat.—This constant has been determined by heating a specimen of known weight in a furnace and when brought to constant temperature allowing it to drop into a calorimeter. The error due to the heat lost in transferring the specimen is usually greater than has been appreciated. The increase in specific heat with temperature appears to represent in general a linear function, though this has not been established beyond doubt and in some cases it is known that the relation is a more complicated one.

Thermal Conductivity.—The methods for determining this constant apparently most in favor at the present time are those involving the measurement of heat loss computed from the input of electrical energy utilized in heating coils surrounded by the material under test.[1] Comparative tests are frequently made by measuring the temperature drop between known or equal distances of different materials, heated from a source of constant temperature and brought to thermal equilibrium. In general, heat conductivity decreases with the porosity of the materials.

Thermal Expansion.—The linear expansion upon heating specimens to known temperatures by immersion in baths maintained at a constant point may be accomplished by direct measurements using a micrometer microscope, by a system of levers, magnifying the expansion so that it may be read from a scale or, through the use of an optical lever, with mirror, scale and telescope. More refined methods, as that of Fizeau, are scarcely ever used in technical work. Probably one of the best known and most satisfactory methods is that of A. W. Gray,[2] using bars 30 cm. long, immersed in an oil bath or heated in an air furnace. The length changes are determined by means of two microscopes clamped to an invar

[1] R. A. Horning, *Trans.* Am. Ceram. Soc., Vol. **19.**

[2] *Sci. Papers* Nos. 332 and 352, Bureau of Standards.

bar, the distance between them being equal to the original length. The readings are made focusing upon fine platinum wires hung from each end of the specimen and extending downward through and below the furnace.

A simple apparatus for determining thermal expansion is illustrated in Fig. 6, consisting essentially of a container provided with a heating cooling and stirrer. The specimen is placed upon quartz glass supports and a quartz rod is placed upon the upper end. A lever, with a mirror attached to its end, is brought in contact with the quartz rod and the initial and final readings made by means of the telescope and scale. For higher temperatures up to about 300°C. "Crisco" may be used in the bath.

Resistance to Temperature Changes.—Although in general it may be said that the ability of refractories to resist sudden temperature changes is a function of the coefficient of thermal expansion, increasing with the decrease of this con-

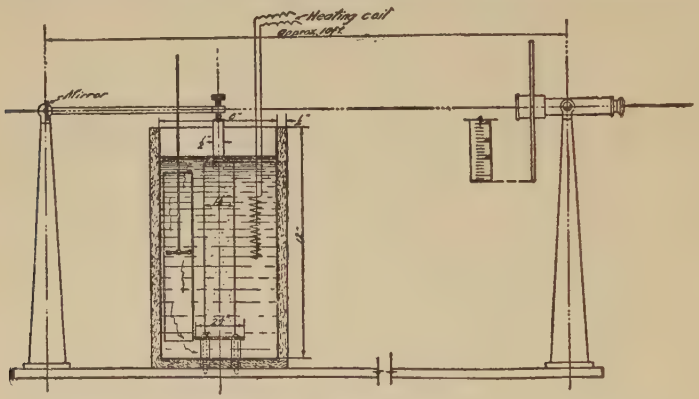

FIG. 6.—Testing thermal expansion.

stant, there are many factors affecting this property. Furthermore, the physical changes brought about by the continued heat effect in use may change the character of the refractory very decidedly with time. Thus, a firebrick may resist sudden temperature changes very well in the condition it comes from the kiln but after having been in place in the furnace for some time it may possibly become denser at the surface, through vitrification or the absorption of slag or glass and thus show a decided tendency to spall. The specimens therefore should be fired to a temperature at or above that of their use before being tested. With some kinds of refractories their tendency to spall when heated or cooled quickly constitutes a serious draw-back to their use.

No standardized tests for the estimation of this quality are available at the present time and the practices vary widely. The specimens are heated to temperatures varying from 800 to 1,000°C. and are subjected to treatments consisting in quenching the test pieces in cold water, either by complete or partial immersion or in subjecting them to cooling in an air blast. After such treatment, varying from 1 to 5 min., the specimens are replaced in the heating furnace and the operation repeated a specified number of times or until failure occurs. The criteria for comparison may be the number of treatments withstood, the weight of material broken off or the ratio between the original and final strength, usually the modulus of rupture. The last procedure would seem to be the most rational.

Resistance to Slagging Action.—According to the porosity of the refractories, the temperature, viscosity and the chemical nature of the slag or glass the absorp-

tion of fused material is more or less pronounced. It is evident that for best resistance the refractory should be dense and as impervious as possible and not subject to chemical interaction with the liquid slag. However, it appears that not only the total volume of pore space but also its capillary character, that is, the fineness of the pores, is of great significance in this respect, as well as whether they are sealed or communicating.

The tests conducted for estimating the quality of resistance to slag corrosion consists usually in cementing a refractory ring to the surface of the test piece and placing within the space of the former a definite quantity of the slag in question. Upon heating the specimen thus prepared to a specified temperature, usually 1,350°C., the absorption of slag is noted by breaking the test piece through the center of the ring and observing the area through which the slag has penetrated. In the tentative specifications of the American Society for Testing Materials the ring is of 2.5 in. inside diameter and 0.5 in. deep. In the case of refractories for use in the glass industry small crucibles are usually made, charged with a glass mixture, more corrosive than the normal composition, and the extent of the penetration noted.

Mechanical Strength.—The mechanical strength of refractories in the cold state varies between wide limits, according to the density and hardness of the materials. Thus clay firebricks may show compressive strengths from 1,000 to 5,000 lb. and moduli of rupture up to 1,200 lb. per square inch. The toughness of refractories, as measured by their resistance to abrasion, likewise varies greatly. Many of the most heat-resisting refractories stand up but poorly under conditions of abrasion. Under certain conditions, as in the top part of blast furnaces, hard and tough materials are required to resist the wearing action of the charge while for other uses this requirement may be neglected. The toughness of refractory bricks may be determined by means of the "rattler," a machine employed for the testing of paving bricks, with the modification that the entire charge consist of the smaller spheres employed in the standard test of the American Society for Testing Materials.

Electrical Resistance and Dielectric Strength.—These qualities are involved only where refractories are used also as insulators as in electric furnaces, stoves, hot plates, heaters of all kinds, spark plugs, insulators used in Cottrell installations for the removal of dust from heated gases, etc. It is necessary that products so used have as high an electrical resistance as possible at the temperature reached and also good dielectric strength where high voltages are employed and there is danger of puncture. Refractories high in fluxes such as feldspar are apt to break down at comparatively low temperatures. In the case of porcelain it was found by Henderson and Weimer that at 275°C. a body fired to cone 9 showed a reduction in the puncturing voltage to one-thirtieth of the initial value at atmospheric temperature.[1]

For the testing of refractories with reference to their electrical conductivity at higher temperatures a cup of 60 mm. outside diameter, 65 mm. height and having a wall 2.5 mm. thick is made from the material in question and fired. This specimen is placed in a suitable furnace. Electrical contact is provided by the use of molten metal on the inside of the cup and its immersion in a shallow metal bath on the outside. An alternating current, 60 cycle, of 500 volts is employed and the current passing through the material measured by means of a sensitive milliammeter or a dynamometer watt meter. The temperature is determined by means of insulated thermocouples immersed in the molten metal on the inside of the cup.

[1] *Trans.* Am. Ceram. Soc., Vol. **13**, p. 469–75.

The results of the resistance measurements are usually plotted as the $\log_{10}R$ against the temperature, in degrees C. The plots as a rule are approximately straight lines expressed by the relation: $\text{Log}_{10}R = a - bt$, where R = resistance of the specimen in ohms, t = temperature (°C.), and a and b are constants. In computing a resistivity factor, based on the dimensions of the cup, the resistance is calculated per cubic centimeter of the material. The criterion used in comparing the merits of each material is the temperature, T_e, at which the material has the definite resistance of one megohm per cubic centimeter, a value used by the electrical division of the Bureau of Standards in evaluating different types of spark plugs.

Types of Refractories.—The several types of refractories may be classified from the technical standpoint as follows:

1. Clay refractories:
   (*a*) High in clay substance.
   (*b*) Low in clay substance, siliceous.
2. Silica refractories.
   (*a*) Lime bonded.
   (*b*) Clay bonded.
3. Aluminous refractories.
   (*a*) High in alumina.
   (*b*) Aluminum silicates of sillimanite type.
4. Magnesite refractories.
5. Dolomite refractories.
6. Refractories of spinel type.
7. Carbon refractories.
8. Silicon carbide refractories.
9. Special refractories.
10. Insulating refractories.

CLAY REFRACTORIES

Clays High in Clay Substance.—The purest type of clay is that represented by carefully washed kaolin consisting essentially of a hydrous silicate of aluminum, $Al_2O_3.2SiO_2.2H_2O$, corresponding to the following percentage composition

	Hydrous, Per Cent	Dehydrated, Per Cent
Silica	46.3	53.8
Alumina	39.8	46.2
Combined water	13.9	

While this substance, commonly called kaolin, does not correspond to the most refractory mineral combination of silica and alumina found in nature, it is at least the most commonly distributed material, since it may be assumed to be the fundamental constituent of all fireclays. Other minerals, such as sillimanite, cyanite, and andalusite, corresponding to the general formula $Al_2O_3.SiO_2$, are far more infusible, but are of comparatively rare occurrence in clays.

The so-called melting point of pure clays is close to that of platinum; that is, about 1,750°C. Substances whose softening temperatures differ too greatly from that of kaolin should not be considered as fireclays. Though the chemical composition of fireclays approaches more or less closely that of kaolinite, $Al_2O_3.2SiO_2.2H_2O$, they differ widely as regards their physical structure, varying through all stages from

the well-defined crystalline state to that of a typical colloid. From the technical standpoint, roughly, three classes of refractory clays may be distinguished, *viz.*, kaolins, flint clays, and plastic clays.

Kaolins.—The first class of materials, usually of geologically primary origin, consists, in the purified state, of white clayey matter, containing both the crystalline and amorphous varieties of clay base. In some of these clays the crystalline constituents predominate, as in the North Carolina kaolins. The plasticity of these clays is but feebly developed, though where the granular matter has been broken down by the action of water or other agencies to the amorphous condition, a fair working quality may result. These clays, on account of their whiteness, are used in the pottery industries.

There are, however, kaolins which possess a fair degree of plasticity, as the Georgia and Florida kaolins and some of the English china clays. These, as long as they maintain good whiteness, are highly valued in the manufacture of white ware and porcelain products. Frequently, however, increased plasticity is coincident with increased content of fluxes and consequent reduction in refractoriness. While marked plasticity in itself, of course, does not mean reduced refractoriness, it indicates geological conditions which tend to incorporate impurities in the clay.

Owing to their purity (absence of basic oxides) the kaolins are the most refractory clays. Thus a well-known kaolin from Zettlitz, Bohemia, which is considered a standard material of its kind, shows a softening temperature corresponding to Seger cone No. 35, estimated to correspond to a temperature of 1,750°C., according to the scale adopted by the Bureau of Standards (palladium = 1,549°C.; platinum = 1,755°C.).

Flint Clays.—The so-called flint clays embrace many materials of a grade of purity corresponding closely in composition to the best grade of kaolins. Like the latter, they may, of course, deteriorate into clays of comparatively low refractory value. As has been said, their composition is very close to that of the kaolins, even as to the content of chemical water. Physically they are unlike the soft and chalky kaolins in possessing a hard, dense amorphous structure, showing a peculiar well-defined conchoidal fracture. The color is usually gray. The initial plasticity is exceedingly feeble, though if exposed to the weather or if ground either dry or wet the condition of colloidal "set" may be partially overcome and sufficient plasticity developed for molding purposes. Owing to the weak plasticity possessed by flint clays, their drying shrinkage when ground and made up with water is very slight. On the other hand, in burning these clays undergo a considerable shrinkage. In this respect, however, considerable difference exists between the various materials, apparently dependent upon the colloidal state of the clay base. Thus, for instance, a Pennsylvania flint clay from Clearfield County showed a linear shrinkage of 5.5 per cent up to cone 20; a Maryland clay of the same type was found to possess a maximum shrinkage of 5.6 per cent at cone 11, which decreased on further heating to 3.5 per cent at cone 20, indicating an increase in volume; a Kentucky flint clay, on the other hand, showed a steady gain in shrinkage up to cone 9, when it reached a linear contraction of 9 per cent, while at cone 20 it showed 9.5 per cent.[1]

The volume shrinkage characteristic of these clays subjects the structure of the product into which they enter to a severe strain, which, owing to the low tensile strength, may cause serious difficulty due to cracking and checking, so that it may be

[1] J. M. Knote, *Trans.* Am. Ceram. Soc., Vol. **12.**

necessary either to calcine the flint clay before incorporating it in the body or to replace it in part by ground waste bricks (grog).

The burning shrinkage in the case of flint clays cannot be entirely attributed to the contraction accompanying vitrification. Considering the purity of these clays it is evident that part of the shrinkage is independent of this factor and must be due to a molecular change of another kind, that peculiar to many typical amorphous

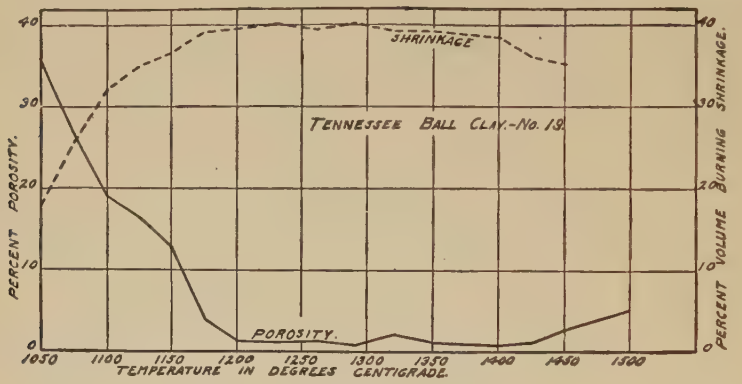

FIG. 7.—Tennessee ball-clay tests.

substances, like alumina, magnesia, zirconia, etc. We may, therefore, ascribe the high burning-shrinkage of flint clays to colloidal volume changes. Geologists are not agreed as to the origin of flint clays, an obscure subject upon which definite information seems to be lacking.

Besides the flint clays proper there are certain dense fireclays of feeble initial plasticity which lack the other characteristics of this class of materials. Such a clay

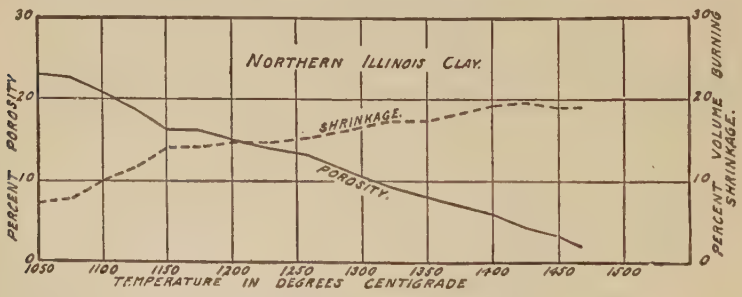

FIG. 8.—Illinois ball-clay tests.

occurs in Alabama, which evidently had been formed under conditions radically different from those producing flint clays. Hard nodules of amorphous silica (flint) were found to be associated with this clay which in itself is dense, of white color and by fine grinding can be made to attain a fair degree of plasticity.

Flint clays occur in extensive deposits in the upper Mercer and Middle Kittanning formations of Pennsylvania and Maryland, the Sciotoville in Kentucky and the Lower Kittanning and Upper Freeport in Ohio as well as in the pocket formations of the pre-Pennsylvania limestones, 40 to 140 miles west of St. Louis, Mo., probably of cretaceous origin. The Ohio flint clays as a rule are not equal in refractoriness to the others mentioned for which a softening point of cone 33 to 35 is demanded.

Plastic Refractory Clays.—Clays combining good plasticity with high refractoriness are not of common occurrence. While there are a number of such deposits in the United States, the majority of these materials show plasticity at the expense of heat-resisting power, that is, they soften at temperatures below that of cone 32. Occasionally they develop in limited quantities as the result of the weathering of flint clay.

The most important sources of such clays are to be found in the so-called ball clays of Paris and Whitlock, Tenn., Mayfield, Ky., Anna, Ill., Enid and Senatobia, Miss., and in the plastic fireclays from St. Louis and southern Ohio. These clays are of great importance as a source of clay for the manufacture of refractories for the glass industry and of graphite crucibles. Plastic clays of good quality are also found in the Golden district, Colorado, Utah and California.

Analyses of several typical clays of this class are given in table:

TABLE 2

	Silica	Alumina	Ferric oxide	Titanium oxide	Lime	Magnesia	Potash	Soda	Loss on ignition
St. Louis	54.64	28.75	3.00	1.85	0.30	0.35	0.15	....	11.00
Whitlock, Tenn., Clay No. 5	47.70	33.28	1.21	not det.	0.45	0.75	not det.	not det.	16.25
Mayfield, Ky., Clay No. 4.	53.22	32.36	1.17	not det.	0.40	0.83	not det.	not det.	12.44
Mayfield, Ky., Clay No. 6.	60.80	24.62	0.88	2.80	0.46	0.13	0.50	0.38	10.36
Klingenberg, Germany	50.76	29.26	1.51	1.61	1.08	0.73	0.85	0.19	14.24

The chemical composition of the German Klingenberg clay has been given for comparison inasmuch as it represents one of the best known bonding clays known, especially for the manufacture of graphite crucibles.

It will be noted that these compositions show a higher silica and lower alumina content than the kaolins and flint clays, and in addition show replacement of the Al_2O_3 and combined water by fluxing constituents, iron oxide, lime, magnesia and alkalis. This replacement when continued beyond the bounds indicated results in further decrease of the heat-resisting qualities until we come to the plastic fireclays of low grade. This makes it necessary to classify the plastic fireclays of this type according to their softening point into No. 1 materials softening at or above cone No. 32, No. 2 clays softening between cones No. 30 to 32, No. 3 clays softening between cones 28 and 26 and No. 4 materials fusing between Nos. 26 and 19. This classification is an arbitrary one and does not hold strictly for siliceous clays as will be shown later.

A large class of fireclays associated with the coal measures of Pennsylvania, Maryland, Ohio, Kentucky, West Virginia and Indiana belong to the types softening between cones 28 to 32. These are useful for bonding purposes in the manufacture of firebricks and shapes. Such clays are available in enormous quantities in conjunction of the coals of these states. All of the plastic bond clays, whether of the No. 1 grade or not, fire to a buff color which may become more and more discolored through the presence of iron pyrites or iron oxide as we descend in the scale of refractoriness.

The purest as well as the strongest clays are probably the so-called ball clays, the plasticity and bonding power of which is reinforced by the presence of organic matter. This type of clay seems to be preferred in the manufacture of graphite crucibles, and other special refractories.

The Testing of Refractory Clays.—In the study of clays it soon becomes evident that the chemical composition is not the only criterion of their value. It is impossible to foretell anything concerning the physical properties of a clay from its composition excepting the refractoriness which can be estimated fairly accurately. For this reason it is evident that tests of the physical properties are of great significance. It might be well to consider some of these together with simple tests devised for their numerical evaluation.

One of the first constants we desire to know about a clay is the amount of water required to impart to it the consistency known as normal consistency. By this we mean the condition in which the clay is neither too stiff so that it cannot be molded and shaped readily nor so soft that the clay will stick to a bright nickel knife or spatula. The experienced operator will hit this condition with considerable accuracy, but unfortunately we have no device for gaging it with absolute accuracy. The Vicat needle used in cement testing has been suggested for this purpose. It is surprising, however, how closely an experienced operator can check his work in obtaining practically the same consistency. In determining this water content, then, we must make up a briquette of the plastic clay, weigh it at once and dry it. The final drying should take place at a temperature of 110°C. The calculation merely consists in subtracting the weight of the dried piece from its weight in the plastic state and dividing by the dry weight of the specimen. This result multiplied by 100 gives the percentage of water required by the clay, in terms of its dry weight. As a rule the higher the required water content of a clay is the more plastic the material is, but if this value becomes too high the plasticity becomes stickiness which is not so desirable.

The quality of clay known as its plasticity or the property of permitting its molding and shaping we have as yet not been able to express numerically by means of test results. All we can do at the present time is to estimate this quality by feel or by some indirect determinations. Thus we have already seen that the amount of water required to render clay plastic is a measure of its plasticity. Similarly, the shrinkage in drying is a criterion of this quality. Other means of estimating plasticity indirectly are the capacity of clays to absorb certain dyes like malachite green, the time required for the dried clay to slake down in water, the tensile and transverse strength of dried bars of clay and the fineness of the clay, with reference to the amount of material which fails to settle in water after standing for some time.

The drying shrinkage is determined either by linear or by volume measurements. Since clay never shrinks uniformly in all directions, the shrinkage determination by volume is much more accurate and reliable. For this purpose the plastic clay is made up into a briquette which is at once immersed in petroleum and the volume of which is later determined by means of the voluminometer. The specimen is allowed to dry and is again immersed in petroleum. After standing for some time so that it is saturated with the liquid, its volume is once more determined. Evidently, the volume in the wet state, minus the volume in the dry state, divided by the dry volume, the result being multiplied by 100 gives the percentage volume shrinkage in terms of the dry volume.

Since in the shrinkage of clays a volume of water equal to the volume of the contraction is evaporated, we may speak of such water as shrinkage water. The volume of water remaining in the clay, after shrinkage has ceased, is called pore water. The relation between the volume of the pore and that of the shrinkage water is interesting and significant.

The volume relations involved may be readily computed from the following relations:

$\frac{100.\ S(W_1 - W_2)}{W_2}$ = Volume of total water in terms of the true clay volume, expressed in per cent.

and

$$\frac{100.\ S(V_1 - V_2)}{W_2} = \text{Volume of shrinkage water, in terms of the true clay volume, expressed in per cent.}$$

In these formulas,

S = Specific gravity of clay, approaching the value 2.60 quite closely for all clays of this type,
W_1 = Weight of briquette in the wet state,
W_2 = Weight of dried briquette,
V_1 = Volume of briquette in wet state, and
V_2 = Volume of dried briquette.

Subtracting the volume of the shrinkage water from that of the total water, we can compute the volume of pore water. The ratio of the pore-water volume to the shrinkage-water volume is characteristic for the several types of clay. The lower this value, the more plastic is the material in question. For bonding clays in general, the ratio should not exceed 1:1.

A simple method for estimating the plastic character of clays consists in grinding them together with potters' flint in the proportion of 1 : 1, making them up to a plastic mass, and forming 1-in. cubes. After drying these thoroughly and immersing them in water, the time required to slake the clay down to a mud is characteristic of the structure of the clay. Heavy, "fat" clays will break down very much slower than lean and less plastic ones.

The mechanical strength of dried specimens of clay prepared from the unmixed material, or from a 1:1 mixture of clay and sand, is another means of estimating indirectly the plasticity and bonding power. This is especially true when the sand mixture is used, owing to the fact that some of the very "fat" clays are apt to check and crack in drying, without the addition of a non-plastic. The tensile strength is determined upon specimens made in the form of cement test briquettes which are broken in a suitable machine. The transverse strength test which is now made more commonly, employs bars 7 by 1 by 1 in. which, when dried, are loaded at the middle and broken. From the dimensions of the bar, the span and the load, the modulus of rupture is determined, expressed in pounds per square inch. The higher the strength, the greater should be the plasticity and the bonding power.

The fineness of the clay with reference to its finest particles, *i.e.*, grains of the magnitude of 0.01 to 0.003 mm., is indicative in general of the content of the dispersed or colloidal portion, but is not necessarily a function of plasticity. Plasticity, therefore, is not merely a matter of fineness. The apparent specific gravity of the dried clay is sometimes useful in comparing different materials.

The Firing Behavior of Clays.—With reference to the behavior of a bond clay when heated to higher temperatures, several points must be noted. These are—the temperature at which the clay first becomes dense or vitrified, the temperature at which it overfires, and the point at which it shows evidence of fusion.

The overfiring point represents that condition at which the clay shows excessive vitrification and softening under pressure. Most clays at this point begin to evolve gases which cause the structure to become vesicular or spongy. This state usually marks the end of the usefulness of the clay by itself.

The firing behavior of a clay is determined by making up a series of briquettes, about 2 by 1¼ by 1 in., placing them in a suitable kiln or furnace and drawing one or more specimens at different temperature intervals. It is necessary, of course, to maintain accurate temperature control by means of a pyrometer or by the use of standard pyrometric cones. It is customary to draw the first specimen at 1,050°C.

and from there on, every additional 25°C. up to 1,500°C. The rise in temperature above 900°C. should be at the rate of from 20 to 25°C. per hour.

Upon removal from the furnace the briquettes are either covered with hot sand or placed in a small furnace kept at a red heat so as to avoid checking or cracking on cooling. When cooled, the specimens are weighed and then immersed in clean water, boiling under a partial vacuum so that the liquid will penetrate into all the open and communicating pores. It is then possible either to make a simple absorption or a porosity determination. For general work the former will suffice. If then the absorption data are plotted against the temperatures either in degrees or expressed in cone numbers a curve will be obtained which shows the firing behavior of the clay very clearly. The vitrification temperature will be that point at which the absorption falls to a minimum, usually a value not exceeding 1 per cent. The overfired condition is indicated by a marked rise in the absorption due to the formation of a spongy or vesicular state. In many of the published tests, the porosity value is plotted against the temperature in place of the absorption. The softening point determination has already been described in previous paragraphs.

There are thus three fixed points to consider in the testing of fireclays or for that matter in the examination of refractories in general, *viz.*, the vitrification temperature at which the mass becomes dense and practically impervious, the point at which it becomes overfired and vesicular and the softening temperature. The difference between the vitrification and overfiring temperature constitutes the vitrification range which is of considerable technical importance from the manufacturing standpoint.

The results of physical tests made upon a number of refractory bond clays are given in Table 2.

With the increase in silica and the presence of varying amounts of fluxes the coal-measure fireclays tend to become dense or vitrify at higher temperatures than the bond clays even though the latter possess a higher ultimate refractoriness. It is evident, however, that there are also many clays of this type maturing at a low temperature, as for instance the carboniferous clays from Brazil, Ind.

For high-grade clays of any kind it is necessary that the total fluxes be as low as possible. From certain work done by the writer it appears that more than 0.22 molecular equivalent of fluxes is undesirable with a silica content up to $Al_2O_3.3SiO_2$. With a higher content of silica the permissible RO content decreases. Thus a clay of the composition 0.17 $RO.Al_2O_3.4\ SiO_2$ is apt under the compression conditions met with to fail at furnace temperature.

Siliceous Clays.—The necessity of dividing the refractory clays into two classes, low and high in silica arises from the fact that the two types behave quite differently in use. The division line between the two kinds of materials has been arbitrarily fixed at a silica content of 70 per cent. It is obvious that the gradation from one class to the other is not an abrupt one, but that the properties of the first gradually merge into those of the second. These clays thus consist of clay substance plus a considerable amount of free silica as quartz sand.

There are several characteristics by means of which the two types may be distinguished. As a class, though with certain exceptions, the siliceous clays possess less plasticity, bonding power and strength in the dry state than the more aluminous ones, and also a low drying shrinkage. In the firing process they usually show no decided decrease in porosity and hence remain porous. As a result, their firing shrinkage is very low and not infrequently *nil*, due to the expansion of the quartz.

Finally, siliceous clay refractories upon reheating in use are free from the tendency to shrink characteristic of the high clay materials. In addition they are able

TABLE 3

Number	Source of clay	Percentage of water content in terms of dry weight	Per cent volume shrinkage in terms of dry volume	Ratio pores to volume of shrinkage water	Time of slaking 1 clay 1 sand, minutes	Apparent specific gravity of dried clay	Modulus of rupture of 1 clay 1 sand, lb. per sq. in.	Temp. of vitrification, degrees Centigrade	Over-firing temperature, degrees Centigrade	Softening temperature in cones
1	English ball clay	44.90	43.30	0.79	30	1.69	323	1,100	1,320	31½
2	Klingenberg, A. T.	39.68	42.58	0.57	78	1.79	381	1,125	1,450	32
3	Klingenberg, E. T.	50.66	55.05	0.56	108	1.70	363	1,100	1,450	32
4	Missouri	43.63	43.63	0.72	33	1.72	351	1,175	1,450	32
5	Kentucky	45.28	34.53	1.05	9	1.55	234	1,260	1,450	32
6	Ohio	25.30	27.92	0.83	9	1.94	309	1,230	1,350	30½
7	Ohio	22.08	24.46	0.80	8	2.01	281	1,290	1,400	31
8	Missouri	34.66	42.63	0.54	117	1.90	554	1,175	1,320	27
9	Illinois	45.16	45.35	0.65	54	1.67	341	1,200	1,450	32
10	Kentucky	50.85	46.80	0.71	37	1.57	362	1,230	1,450	32
11	Tennessee	44.73	35.27	0.97	27	1.56	282	1,175	1,450	32
12	Maryland	38.91	40.61	0.67	114	1.76	518	1,175	1,390	31
13	Illinois	40.98	38.81	0.75	45	1.66	262	1,285	1,475	32
14	Mississippi	31.39	31.36	0.81	55	1.81	326	1,290	1,450	30
15	English ball clay	40.33	39.85	0.75	41	1.71	389	1,100	1,425	32

Similar results for fireclays of the No. 2 and lower types are reported in Table 4.
(H. G. SCHURECHT, *Jour. Am. Ceram. Soc.*, Vol. **1**, 267–86.)

TABLE 4

No.	Source	Per cent water in terms of dry weight	Per cent shrinkage in terms of dry volume	Apparent specific gravity of dried clay	Modulus of rupture of clay dried at 110°C.	Shrinkage at max. firing contraction	Temperature of vitrification in cones	Softening point in cones
1	Roseville, Ohio	27.5	30.8	2.08	218.6	17.12	14+	26+
2	Mogadore, Ohio	19.1	16.1	2.02	142.5	14.10	14+	18
3	N. Brighton, Pa.	21.0	20.0	1.99	194.2	12.43	14	28
4	Crooksville, Ohio	24.3	22.7	1.98	178.8	13.70	14+	28
5	Nelsonville, Ohio	23.0	22.7	1.89	325.0	13.30	14+	31

to withstand pressure conditions even at high temperatures, a property of great value for many uses. It is this rigidity which justifies their use in such installations as the benches carrying gas retorts, furnace crowns, etc.

The degree of fineness of the quartz is quite important in this connection. Coarser grains are undesirable and are apt to give rise to "checking" or cracking due to unequal expansion.

The firing behavior of a siliceous clay as compared with one of the high clay type is illustrated in Figs. 7 and 8, representing respectively a Tennessee ball clay and a siliceous clay from near Ottawa, Ill. It will be observed that the former reaches

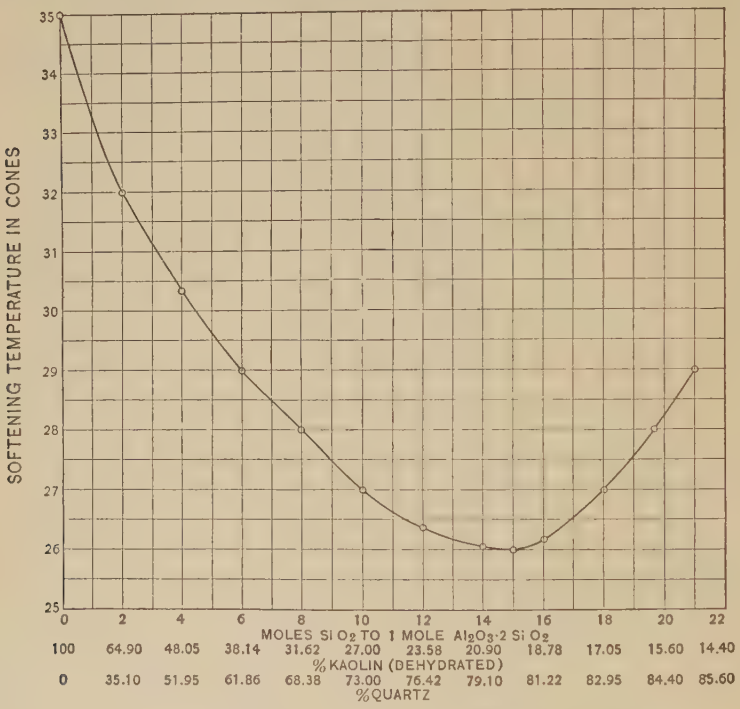

Fig. 9.—Seger's softening-temperature diagram.

complete maturity at 1,200°C., undergoing at the same time a contraction in volume of 40 per cent while the latter does not attain minimum porosity even at 1,475°C. with a volume shrinkage of only about 13.5 per cent, beginning with an initial temperature of 1,050°C. Other siliceous clays show these characteristics in a still more marked degree.

It was realized early in the study of fireclays that any addition of free silica to pure clay substance lowered the softening temperature. Thus Seger found ("Collected Writings," p. 434) that a mixture corresponding to the molecular formula $Al_2O_3.17SiO_2$, composed of kaolin and ground quartz softened at cone 26, while the clay itself fused at cone 35. This composition represents the eutectic between the two components, consisting of 23 per cent of hydrous clay substance and 77 per cent of quartz. Seger's eutectic composition agrees as a whole with the more accurate determinations of Shepherd and Rankin. The curve of Seger is given in Fig. 9.

Siliceous clays therefore are inherently less refractory than the more aluminous ones and at the same time they are more sensitive to the presence of basic fluxes.

Unless these clays are low in such constituents they are apt to fuse at too low a temperature. If care is used to exclude the more impure siliceous clays this does not detract from their usefulness as refractories where rigidity at furnace temperature is a prime consideration.

Analyses of typical siliceous fireclays are:

	Lester, Ark. Per Cent	Gross Almerode, Germany, Per Cent
Silica	72.05	73.08
Alumina	17.21	15.75
Titanium oxide	.....	2.10
Ferric oxide	1.25	0.62
Lime	0.45	0.54
Magnesia	0.75	0.33
Potash	.....	0.27
Soda	.....	0.16
Loss on ignition	8.56	7.11

Refractory Clay Products.—From the standpoint of the manufacturer the clay refractories may be subdivided into the following divisions: (1) Flint-clay refractories; (2) siliceous-clay refractories; (3) plastic-clay refractories; (4) grog refractories.

Flint-clay Refractories.—The flint-clay refractories consist essentially of a predominating amount of flint clay, as high as 85 per cent in the best qualities, with a smaller admixture of plastic fireclay, usually just sufficient for bonding purposes. In Europe no flint clays are available and hence clays of the plastic type as well as kaolins are used exclusively. It is important to use as small an amount as possible of plastic clay in the bonding of flint clay except when the former is of exceptional quality, since after all it is the small amount of more fusible material corresponding to the eutectic which governs the refractoriness of the whole mass. It would be desirable in certain instances to grind together the plastic clay with part of the flint clay so as to increase the refractoriness of the bonding material. An ideal combination would consist of flint clay bonded together with weathered plastic clay of the same type, but usually such a material is not available. Another very satisfactory composition would be a mixture composed of flint clay and plastic kaolin, like the somewhat impure top clay overlying the kaolin deposits of Georgia.

The heat-resisting qualities of this class of refractories are decreased, of course, with decrease in flint clay and mixtures containing as low as 20 or 30 per cent of this material and the balance of plastic fire clay cannot be expected to be more refractory than the latter. In the case of flint clays which show a large contraction in firing it may be necessary to calcine a portion, either in the regular down draft or in shaft kilns, but the practice is avoided as much as possible in the making of standard bricks. For tiles and larger shapes, however, the use of some calcined "hard" clay is often necessary.

Firebricks containing much flint clay are manufactured by grinding the clay mixture in dry pans, screening and tempering or pugging it with water in a pug mill or wet pans. The bricks are molded either by hand or in so-called soft mud machines. It is possible also to press such mixtures into bricks in the damp state, using for this purpose powerful presses. Bricks thus made are said to be "dry-pressed" and are considerably heavier than those made from the soft-plastic mixture. They also possess physical properties decidedly different from those of the soft mud brick,

resulting in greater resistance to compression at furnace temperature but lessened resistance to sudden temperature changes and to slag action. Mixtures composed of a smaller proportion of flint clay and more of the plastic can be made molded into bricks by means of auger or piston presses. Shapes are almost invariably hand molded.

The firing of the refractories is usually conducted in round, down draft or beehive kilns and the temperature reached may vary from that of cone 5 to cone 10 or 11. There has been a tendency in the past to carry the burning to a point not sufficiently high, with the result that the bricks shrink in use more than they should. However, this practice is being eliminated more and more and it is being realized that for general purposes the product should be fired as high as possible. The more economical continuous regenerating kilns are being used only in a few plants.

Properties of Flint-clay Refractories.—These may be briefly summarized as follows: The refractoriness of No. 1 materials as indicated by the softening point is usually not lower than that of cone 31 (about 1,685°C. or 3,065°F.). Kanolt has found the mean melting point of 41 samples of firebrick to be 1,649°C., determined in the Arsem furnace.

The permanent contraction or expansion of such products when heated uniformly in a suitable furnace to a temperature of 1,400°C., maintained for 5 hr., is as a rule not more than 1 per cent in terms of the original length. For high-grade materials the allowable contraction should not be more than 1 and the expansion not more than 0.5 per cent. For materials used for less severe heat duty softening temperature may be dropped to cone 28 and the reheating temperature to 1,350°C. The porosity of these products varies as a rule from 20 to 30 per cent and the specific gravity from 2.5 to 2.7.

Products of first quality should be able to resist a pressure of 40 lb. per square inch at a maximum temperature of 1,350°C., with a compression of not more than 5.5 per cent in terms of the original length.

The specific heat possesses a value close to 0.2. According to Mellor the specific heat of firebrick varies with temperature according to the relation: $S = 0.193 + 0.00006t$, where S = mean specific heat and t = temperature in degrees Centigrade.[1] The thermal conductivity K,[2] according to Heyn increases with temperature (K gives when divided by 100,000, cal. per second through 1 cm.[3] per 1°C., difference in temperature between the faces). Thus for 200°C., $K = 145$; for 400°, $K = 180$; for 600°, $K = 220$; for 880°, $K = 240$; for 1000°, $K = 260$; and for 1100°, $K = 270$. In general the conductivity varies inversely with porosity though the size of the pore spaces is manifestly of considerable importance. It would be expected also that this property is altered by reheating. From data published by the Celite Products Co. it appears that the thermal conductivity of average flint clay fire brick is about 0.002064 cal. per cubic centimeter per second, per degree Centigrade. at 149°C.; 0.00275 at 538°C. and 0.003784 at 1,037°C. Expressed in B.t.u., per hour, per square foot, through 1 cu. in. and per degree Fahrenheit temperature difference the heat conductivity would be approximately 5.8, at 200°F., 7 at 600°, 9 at 1,200° and 12 at 2,200°F.

Horning[3] reports the B.t.u. transmitted per 24 hr., through 1 sq. ft. and 1 in. thickness per degree Fahrenheit temperature difference, to be 92 at 300° F. for dense and 62 for porous firebrick. At 1,000°F. the same value is 116 for the dense and 82 for the porous brick.

The data available relating to thermal expansion are meagre and where given no statement is made as to the character of the firebrick in question, whether porous or

[1] *Trans.* English Ceram. Soc., Vol. **12,** p. 279.
[2] *Stahl u. Eisen,* Vol. **34,** p. 832.
[3] *Trans.* Am. Ceram. Soc. Vol. **18,** p. 192.

dense, whether high in clay or siliceous. This constant is likewise subject to variation both with temperature and structure changes in the material. The value is of the magnitude of 0.0000008 per degree Centigrade, being lower for porous and higher for dense materials.

High-grade flint fireclay products resist temperature changes well but this quality diminishes rapidly with the content of plastic bond clay and the resulting density. As a class these refractories do not resist slagging action well owing to their porosity. This quality is improved either through the increase of plastic clay or by very hard firing.

The mechanical strength of these materials differs widely but as a rule the compressive strength does not exceed 1,500 lb. per square inch. Bricks of a lower strength than 1,000 lb. per square inch are indicative of inferior bond and hence should not be used. Little is known concerning the electrical resistance of these refractories at furnace temperatures.

Siliceous-clay Refractories.—Owing to the fact that the siliceous clays possess a greater degree of plasticity than the flint-clay mixtures they may be made into bricks in the plastic state by means of the auger machine or piston press. They are fired to about the temperature of cone 8.

From previous considerations it is apparent that this type of refractory cannot be as heat-resisting as the previous class. However, for first-class materials the softening point should not be lower than that of cone 28, about 1,635°C. or 2,975°F. The permanent contraction or expansion upon reheating to 1,400°C. is, as a rule, low and should not exceed ± 0.5 per cent. The porosity may vary between 20–28 per cent. The resistance to compression at temperatures of 1,350°C. is high and the contraction should in no case exceed 4 per cent at the temperature given and under a load of 40 lb. per square inch.

No reliable information is available concerning the thermal properties of these refractories but it is known that their conductivity is somewhat greater than that of the flint-clay refractories. This applies also to the coefficient of thermal expansion. Siliceous refractories do not resist sudden temperature changes well. They also cannot be expected to resist the action of slags except the latter be siliceous in character. The compressive strength is about that of the flint-clay brick but the resistance to impact and abrasion is lower.

Plastic-clay Refractories.—These materials are usually made into bricks by means of the auger machine and finally fired to temperature between the softening points of cones 4 to 8. The so-called fusion point of these materials should not be lower than that of cone No. 26, about 1,600°C. or 2,912°F. No load-resisting requirements are made upon refractories of this type but upon reheating to 1,290°C. (2,354°F.), maintaining this temperature for 5 hr., they should not show a contraction of more than 1.5 per cent of the original length nor an expansion of more than 1 per cent. These refractories are much denser and harder than those of the previous classes.

The porosity may often be as low as 10 per cent. Their thermal properties may vary widely and both the conductivity and expansion are greater than those of the higher grade refractories and often as high as 10 B.t.u. per hour, through 1 sq. ft. and 1 in. thickness, per degree Fahrenheit and at 1,400°F. Their resistance to sudden temperature changes is certain to be less than that of the flint-clay refractories. Their resistance to corrosion is limited by the softening temperature. Where the metal or slag is well within the safe limit of the material the resistance is usually good and the shapes used in ladles for handling molten metal are often made from this

kind of material. Where the time of exposure to the liquid mass is but short the comparatively low refractoriness of the lining is not of great significance. The mechanical strength of the plastic fireclay bricks is high and frequently the resistance to compression attains 5,000 lb. per square inch and more.

Grog Refractories.—In ceramic usage the term "grog" implies calcined clay or broken and ground waste product, rejected during the course of manufacture due to some defect like cracking, or from material which has already been in use, such as broken firebricks, glass pots, etc. The grog refractories, composed of burnt and plastic clay, generally in the proportion of 1 : 1, constitute a large and important class of materials, especially in the glass industry. In this class of bodies it is necessary to consider not only the character of the clay but that of the grog as well, together with the coarseness or fineness of the latter, expressed by its surface factor, *viz.*, $\Sigma\ a_1/d_1 + a_2/d_2 + a_3/d_3 + \ .\ .\ .$, where a_1, a_2, etc., equal percentage by weight of each group of sizes, obtained by separation through sieving, and d_1, d_2, d_3, etc., equal diameter of each group, determined by means of the micrometer microscope or other methods. The factor thus expresses the total surface of the grains and hence the fineness, and rapidly becomes larger for particles which pass the 80-mesh and finer sieves. It obviously is the more accurate the closer the size groups approach each other, but for practical purposes sizing between the 8 to 20 mesh, 20 to 40, 40 to 60, 60 to 80, 80 to 120, and through the 120-mesh sieve, is sufficient to establish the surface or fineness factor. The finer the particles are the more they will unite with the bond clay, thus increasing its refractoriness if the grog is more refractory, or decreasing it if the latter is deficient in this respect. The finer the grog, the more plastic will be the fired body mixture and the denser and stronger the resulting product. The strength in the dry state increases with grog down as fine as that passing the 100-mesh sieve, but decreases with material passing the 200-mesh sieve. For bodies which are to be strong, dense and resistant to the action of fluxes, fines passing the 80-mesh sieve are necessary up to about 50 per cent, but for products which must resist the shocks of sudden temperature changes the fine material must be kept at about 25 per cent.

Clays high in clay substance as well as siliceous ones are used as bonding materials for grog mixtures. Thus for highly refractory muffles, boiler tiles, etc., a clay of the first type is employed; for zinc muffles, etc., one of an intermediate silica content; while for glass pots, tank blocks, etc., either a mixture of the first and a siliceous clay is preferred or the exclusive use of the last type. A discussion of glass house refractories may bring out facts applicable to other grog bodies.

Glass Pot Mixtures.—It has been assumed by American glass manufacturers that the refractories used by them, glass pots, tank blocks, floaters, boots, etc., for best results require the use of German clay, principally that obtained at Grossalmerode, the properties of which are discussed in preceding pages. The fact that this clay possesses good plasticity, safe drying qualities, low drying and low fire shrinkage, and at the same time, owing to its siliceous nature, resists the action of the commercial glasses very satisfactorily, has made it a standard raw material. The low temperatures employed in American pot arches have required the use of clays of low fire shrinkage, since evidently it is not admissible to permit the contraction taking place in the melting furnace to be too great, owing to the severe stresses to which the pots would be subjected.

Refractories for the glass industry are made, as a rule, from a mixture of calcined clay, consisting partly of crushed old pot material, carefully cleaned, and plastic bond clay. The proportion of calcined (grog) to plastic clay usually is between 1 : 1 to 3 : 2. In some glass plants no clay but the Grossalmerode material and pot shell was used; in others the plastic clay from Missouri was introduced, and, again, in some cases a smaller amount of raw flint clay was added. Some of these mixtures are as follows:

TABLE 5

	Per Cent	Per Cent	Per Cent	Per Cent
Grossalmerode clay, raw	47.0	20	24	25
Grossalmerode clay, burnt	....	10		
Missouri plastic clay, raw	....	20	16	20
Missouri clay, burnt	26.5	..	15	15
Flint clay, raw	....	..	15	10
Pot shell	26.5	50	30	30

These mixtures, taken as a whole, gave satisfactory results with the usual soda-lime glasses. In general, it was not realized that different refractories are required for different types of glasses, and unfortunately no special efforts were made, excepting by one or two large companies, to improve matters. With the elimination of German clay more or less confusion occurred, and the exclusive use of domestic clays was not always successful, for reasons which will be considered later. A search was made for substitutes, with the result that a number of new clays were brought on the market, and the mining operations of the well-known and reliable Missouri clays from the St. Louis district enlarged. Among the clays brought forward in this connection were the ball clays of Tennessee and Kentucky, already used to a large extent in the pottery and tile industries, the siliceous clay from Lester, Ark., the plastic fireclays from near Portsmouth, Ohio, and the clays from southern Illinois. The Arkansas clay was found to be very similar to that from Grossalmerode, but it required to be supplemented by a plastic, more aluminous clay, firing to a dense structure at a lower temperature, approximately 1,250°C. For certain glasses satisfactory results have been obtained with the use of only the Missouri washed plastic clays in the raw and burned state, employing in addition about 10 per cent of calcined flint clay. In other cases, using some of the other clays mentioned, unsatisfactory results have been obtained, due principally to cracking of the pots around the bottom. The resistance of the American clays to corrosion has usually been very satisfactory. The cracking must be ascribed merely to the greater firing shrinkage of the American clays.

Fire Shrinkage.—The differences in contraction of different clays are shown in the following table:

TABLE 6

Clay	Drying shrinkage, in percentage of dry volume	Firing shrinkage, in percentage of volume in dried state, at temperatures of—				
		1,050°C.	1,100°C.	1,200°C.	1,290°C.	1,400°C.
Grossalmerode	20.57	.....	4.14	7.84	8.55	10.70
St. Louis district	28.52	8.70	13.62	19.60	23.25	24.35
Tennessee ball clay	29.51	17.90	32.20	39.12	42.75	38.45
Kentucky ball clay	34.53	11.49	26.80	37.01	41.75	39.55
Southern Ohio, washed	40.61	7.08	14.50	20.20	26.02	25.41

These figures show the point brought out with unmistakable clearness. It will be noted that the fire shrinkage of the German clay is far less than that of the American bond clays.

Assuming that the mean pot arch temperature is 1,050°C., which is above the point usually reached, and that the mean furnace temperature is 1,290°C., the Grossalmerode clay will contract about 7 per cent of the dry volume; the St. Louis clay, 14.55; the Tennessee ball clay, 24.85; the Kentucky ball clay, 30.26; and the Ohio material, 18.94 per cent. Thus, allowing for a grog content of 50 per cent, the contraction in the furnace of a pot having a solid content of 10 cu. ft. would vary from 0.75 to 1.5 cu. ft. of displacement, a condition which taxes the strength of the body to the limit. The figures given show also one reason why the Missouri clays have been preferred to pot making, their fire shrinkage being the lowest of the American clays cited. It is evident, therefore, that the use of a siliceous clay like that from Arkansas with a fire shrinkage of only 1.6 per cent at 1,290°C. is greatly to be desired. Its exclusive use, however, is not desirable, due to its too open structure, and its combination with a clay of the ball clay type should prove eminently satisfactory.

The large furnace shrinkage of American bond clays may be overcome by admixture with a siliceous clay or sand, by increasing the content of grog or by sizing, and by the employment of higher prefiring furnaces.

The use of siliceous clays probably offers the easiest remedy, since materials of this type are plentiful and are to be found in large quantities in New Jersey, as well as associated with many of the fireclay deposits of Missouri, Kentucky, Tennessee, Ohio, Pennsylvania, and other States. It is not desirable to reduce the shrinkage of the clay or the pot body to its lowest possible value; but the fire contraction of the blended clay mixture (without grog) should be about 12 to 15 per cent, in terms of the dry volume, at 1,300°C. It is obvious that exact limits cannot be given to cover all cases.

For the purpose of bringing about a high degree of density and imperviousness the addition of feldspar to the body has been introduced by Scholes, a practice which has met with good success in glass-melting pots. For such uses as zinc retorts the Missouri bond clays with grog from the same source, or sometimes calcined flint clay, are largely used. In all cases the bond clay used must be sufficiently refractory and must not have a softening point below that of cone 31, excepting in the case of siliceous clays which may go as low as cone 28. In addition they must have the required degree of plasticity, and strength in the dry and the fired state.

Special Refractories of the Porcelain Type.—For certain purposes, as the melting of optical glass, of fine enamels, color fluxes, etc., where the presence of iron is undesirable, white bodies may be prepared through the use of grog prepared from a mixture of 80 per cent kaolin, 10 per cent ground quartz and 10 per cent of feldspar. This grog may be replaced by the use of waste pottery bisque (unglazed porous table ware). Mixtures of this type are as follows:

	a	*b*
Kaolin grog	40	
White ware bisque	..	48.0
Raw kaolin	22	23.5
Plastic bond clay	23	22.0
Feldspar	7	6.5
Ground quartz	8	

These and similar mixtures should be brought close to vitrification in order to develop their resistance to corrosion to the maximum where this quality is desired.

Grog clay refractories are almost invariably made by hand with or without the use of molds, though recently the manufacture of pots and other products by the process of casting in plaster molds has been begun. This method consists in preparing a viscous suspension of the clay and grog in water, to which carbonate and silicate of soda have been added in amounts up to 0.4 per cent of the dry weight of the mixture. The best proportion of these reagents seems to be 3 parts of the carbonate to 4 parts of the silicate. These alkaline reagents cause the water content to be not more than that of the plastic mass. The fluid mixture, called slip, is then poured into plaster molds with or without a core, where it is allowed to remain until the desired thickness of wall has formed when the excess is removed. In core molds the casting is solid and the core must be removed before the mass begins to shrink.

Uses of Clay Refractories.—Some of the uses of clay refractories may be indicated as follows: High clay materials, high grade: Blast-furnace linings and stoves, heating furnaces, open-hearth furnaces, soaking pits, boiler installations, stoker arches, muffle furnaces, crucible furnaces, cupolas, malleable-iron furnaces, rotary cement kilns, enameling furnaces, side walls of glass tanks, pot furnaces, carbon furnaces, ceramic kilns, gas producers, sintering and roasting furnaces, gas installations, regenerators, recuperators, reverberatory furnaces, lime kilns, bullion, dross, brass, lead, zinc and copper furnaces, etc.

Intermediate and low-grade clay refractories: Annealing furnaces, runner brick, sleeves and nozzles, ladles, oil refineries, stills, incinerators, vaults, flues, furnace stacks, digesters, bake ovens, dryers, carburetors, etc.

Siliceous refractories, high grade: Beehive coke ovens, gas benches, furnace arches, boiler installations, ceramic kilns, regenerators and many uses given for high-grade materials.

Grog refractories: Glass pots, tank blocks, glass-house accessories, muffles, special shapes for all kinds of furnaces, zinc retorts, etc.

SILICA REFRACTORIES

Lime Bond Refractories.—The materials of this class consist largely of crushed quartzite bonded together with a small amount, usually about 2 per cent, of CaO in the form of hydrated lime. For some purposes blocks of the natural quartzite or quartz schist are employed. There are three principal sources of quartzite suitable for this purpose in the United States, the Tuscarora or Medina formation in Blair and Huntingdon Counties, Pennsylvania, the Baraboo formation, in the Devil's Lake Region, Wisconsin, and the Weisner formation (Lower Cambrian), near Birmingham, Alabama. In addition, quartzite is obtained for this purpose in Montana, from the Quadrant formation (Philipsburg, Mont.). In certain localities, as at Pueblo, Colo., and in Fayette County, Pennsylvania., quartz sandstones are used, and attempts have been made to employ Indiana chert, from the Mitchell Limestone formation.

The quartzites to be suitable for this purpose must be of a character which permits of the transformation of the quartz to cristobalite under heat treatment of reasonable duration and which must retain a structure of sufficient mechanical strength. Both requirements are essential. Materials like chert transform quite readily to cristobalite but are deficient in final strength. Quartzites which fail to show a transformation rate, commercially applicable, or are lacking in strength, must be rejected for this purpose. Although all silica refractories show an increase in permanent volume after reheating this further expansion must be kept within definite limits, $\frac{3}{16}$ to $\frac{1}{4}$ in. per foot or 1.6 to 2.1 per cent of the original length.

The reaction consisting in the inversion of quartz to cristobalite involves a linear expansion of 4.35, that of quartz to tridymite, 5.3 per cent. Although tridymite would thus appear to be the end product, it is not feasible under commercial conditions to carry the inversion that far. We must ordinarily be satisfied to obtain 70 per cent of cristobalite and a small amount of tridymite. In the transformation of quartz to cristobalite the density drops from 2.65 to 2.33, equivalent to a volume increase of 13.6 per cent. The change to tridymite reduces the density of quartz to 2.27, with a volume increase of 16.8 per cent. The manufacture of silica bricks consists essentially in converting the crushed quartzite, with the aid of lime, to a structure consisting essentially of about 70 per cent cristobalite, 25 of quartz, not yet inverted, and 5 per cent of tridymite. The after expansion is hence principally due to the residual quartz. From the standpoint of constancy in volume it would be desirable if the tridymite form predominated since the cristobalite itself is subject to a considerable volume change between 225 to 275°C., from the alpha to the beta modification which has a bearing in the spalling of silica bricks. Upon refiring silica refractories and in use, the amount of tridymite increases steadily at the expense of the cristobalite and quartz until the whole mass may consist of the crystalline constituent in question.

Sandstones, as a rule, invert too slowly. Cherts are transformed rapidly even to tridymite but show poor mechanical strength owing to the general cracking of the particles.

The manufacture of silica refractories consists in the grinding of the previously crushed quartzite in a wet pan, the addition of the slaked lime and the molding of the prepared mixture by hand, in sanded steel molds. The pieces are then dried in tunnels and fired in down-draft kilns for about 9 days, carrying the temperature to the softening point of cones 16 to 18. The cooling requires from 5 to 7 days. The tunnel-car kiln is used in at least one plant, with a maximum heat treatment equivalent to cone 12 to 14.

Properties of Silica Refractories.—The chemical composition of a typica material of this kind made from Pennsylvania quartzite is as follows:

	Per Cent		Per Cent
Silica	96.25	Lime	1.80
Alumina	0.88	Magnesia	0.14
Ferric oxide	0.79	Alkali	0.39

The softening point of the best silica materials is about that of cone 32; it should not fall below that of cone 31.

The permanent linear expansion in the well-burnt product is not more than 2 per cent of the original length when the specimen is reheated to 1,500°C. In the use of silica bricks expansion joints 0.25 in. per foot must be provided. The porosity of silica bricks is usually between 24 to 27 per cent. The specific gravity for material made from Medina quartzite is usually below 2.38 and that from Baraboo quartzite below 2.42. These values may be used in specifications to insure the proper degree of firing.

Silica bricks withstand load conditions satisfactorily even at temperatures up to 1,500°C., under a pressure of 50 lb. per square inch. The specific heat of silica refractories varies from 0.2 to 0.266, between 0°C. and 1,200°. The thermal conductivity, in calories per second, through 1 cm.^{3} per degree Centigrade temperature difference, was found to be 0.0021, for the temperature range 0 to 100°C., and 0.0031, for 0 to 1,000°C. Dudley calculates the mean conductivity between any two temperatures, t_1 and t_2, from the relation: $K = 0.0020 + 0.0000011\ (t_1 + t_2)$.[1] Stockman and

[1] *Trans.* Am. Electrochem. Soc., Vol. **27**, p. 285 (1915).

Foote found the total thermal expansion of a silica brick to be 0.7 per cent of the original length at 600°C., 0.9 per cent at 90° and 1.1 at 1,200°C.

Silica brick do not resist sudden temperature changes well but the harder they have been fired the better they behave in this respect. Refractories containing more tridymite should be expected to stand up better. As a class silica refractories show a far greater tendency to crack or spall upon chilling or rapid heating than clay materials.

Their resistance to slagging action as a whole is not great, owing to their high porosity and acid character. The compressive strength of silica bricks may vary from "2,000 to 5,000 lb. per sq. in." and their modulus of rupture from 500 to 1,500. A modulus of 500 lb. per square inch should be specified for the well-burnt product. Silica brick are used largely in the construction of by-product coke ovens, gas plants, the arches of open-hearth and glass furnaces.

Clay-bonded Silica Refractories.—Ground quartzite rock may be bonded also with clay in place of lime and refractories thus made are frequently known as "quartzite." The amount of plastic fireclay required to bond together the quartzite grains is considerably greater than that of lime used for the regular silica brick, with the result that the softening point is depressed very decidedly according to Seger's curve and especially is this the case if the clay contains fluxing impurities. Thus, considering the melting point of pure silica to be 1,710°C., a rapid drop in fusibility with increasing clay content to that of the eutectic mixture is to be expected and below this point proportionately with the content of basic fluxes present. The eutectic mixture consists of approximately 80 per cent of free silica and 20 of anhydrous clay. The physical properties of this type of refractories have not been established with any degree of accuracy. Mixtures of ground quartzite or ground silica brick and 15 to 35 per cent of clay are commonly used as mortar for laying up silica bricks.

As a class the clay-bonded silica refractories are therefore inferior to the lime-bonded material as regards refractoriness and are not superior to the latter in strength, ability to resist load conditions, resistance to sudden temperature changes, etc., but they usually show a lower after-expansion and sometimes no changes upon reheating. Their use has been principally in the construction of beehive coke ovens, brick kilns, annealing ovens, etc. They should not be made part of large structures like by-product coke ovens as their expansion is too irregular.

Aluminous Refractories.—It can be readily seen that alumina itself, owing to its high fusion point, 2,050°C., is inherently an excellent refractory. The only difficulty in the way of utilizing it is its large contraction upon heating.

This makes it necessary to condense it through the use of the electric furnace by sintering or fusion. The point at which alumina, introduced as bauxite or diaspore, condenses to minimum volume or approximately to this point is necessarily a function of the content of impurities, principally of silica and iron oxide. The purer the raw material the higher must be the sintering or fusion temperature. The fusion diagram of the system SiO_2.Al_2O_3 as given by Rankin and Wright, Geophysical Laboratory, is shown in Fig. 10. Refractories consisting principally of beta alumina may be prepared either from the calcined material alone through fine grinding of part of it or by admixture with some plastic clay as a bond. It is evident that the introduction of the latter will at once lower the fusion point according to the part *ED* of the fusion curve.

The fusion temperature of the eutectic between Al_2O_3 and the next compound, Al_2O_3.SiO_2, is 1,810°C. The use of less refractory clay will lower this point still more and if the amount added is at all large, it will be the determining factor in the deformation under the effect of heat. Aluminous refractories should therefore contain but

little bonding clay and none if possible, by depending upon the use of organic binders and the sintering together of the particles upon reheating the molded articles.

Refractories of this type show considerable deformation under pressure at higher temperatures and hence are not suited for use under these conditions, even though the fusion point is high in itself, approaching 2,000°C. Fused alumina products consist principally of laboratory ware and furnace parts though various larger shapes are also made for heat-resisting purposes. Refractory cements for imbedding heating coils are made from fused alumina and clay. Bricks made from bauxite with some admixture of clay have been found by Kanolt to fuse at 1,820°C. The specific gravity of "alun-

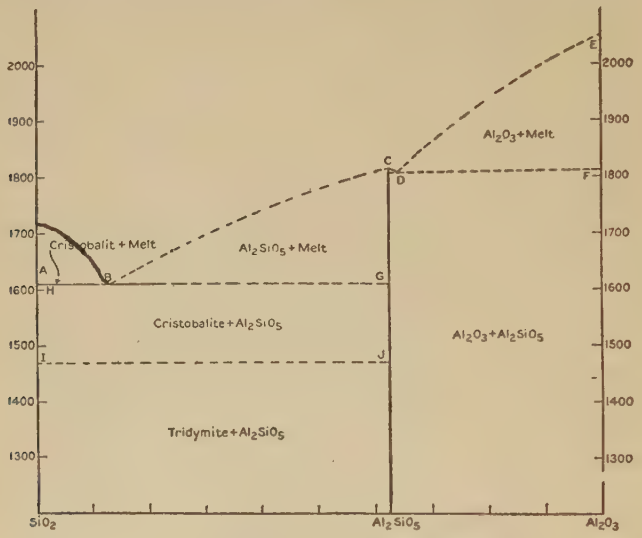

Fig. 10.—Silica-alumina fusion diagram.

dum" ware is given at 3.91, the specific heat at 0.19 at low temperatures, the thermal conductivity at 2.1 times that of firebrick and the coefficient of expansion as being 0.0000071 per degree Centigrade.

Sillimanite Type.—As the silica content of alumina mixtures increases the fusion point decreases until the formation of a compound is indicated at the point *C* of the curve. This point corresponds to the composition $Al_2O_3.SiO_2$ of the mineral sillimanite and to a temperature of 1,816°C. Such mixtures may also be prepared by blending kaolin and anhydrous alumina in the proportion, by weight, of 258 :102 and sintering or fusing this material, in a forced draft or electric furnace. Sintering of the mass down to practically constant volume can be accomplished by heating to about 1,525°C. The addition of 2 per cent of boric acid or of an equal amount of magnesia makes it possible to reduce the temperature to approximately 1,480°C. Microscopic examination should prove the mass to be largely crystalline sillimanite. By suitable grinding and the use of organic binders, such as dextrin, gum tragacanth, linseed oil, etc., the calcine may be worked up into suitable shapes which must be refired to at least 1,450°C.

The product thus made possesses a well-defined melting point, approaching 1,816°C., according to the degree of purity, and is able to resist considerable pressure at furnace temperatures with but little deformation.

Refractories of this type, made from bauxite and fireclay, or from diaspore and clay, have been made by several firms for a number of years and have yielded good results when properly prepared and fired sufficiently high. It is a fact, however, that neither the grinding nor the firing have been carried sufficiently far to give really a sillimanite product. The melting point of bricks made from bauxite-clay mixture has been found by Kanolt to be 1,795°C. It is quite probable that commercially no deliberate attempt is made to arrive at the composition $Al_2O_3.SiO_2$.

Certain refractories like the Marquardt porcelain used for pyrometer tubes consist essentially of sillimanite and sufficient feldspar to bring about a dense structure at about cone 18. Such a mixture would consist of a calcine fired to cone 20 having the composition 71.66 per cent kaolin and 28.34 of anhydrous alumina bonded together by clay. The mixture then would be as follows:

	Per Cent		Per Cent
Sillimanite (calcined)	55	Alumina (calcined)	10
Ball clay (raw)	12	Feldspar	10
Kaolin (raw)	13		

When necessary the plasticity of the mixture can be improved through the use of more clay or increase of the total content of raw clay.

Magnesite Refractories.—The mineral magnesite, $MgCO_3$, is the source of basic refractories used as furnace linings for various metallurgical processes, in the form of bricks or in the crushed state as a mortar and cement for furnace bottoms. The most prominent uses of these materials are in the basic open-hearth furnace and electric-furnace linings.

Both crystalline and crypto-crystalline magnesite are used in this connection, the former having a specific gravity of 3.02, the latter of 2.9–3.0. The most important sources of this mineral are Austria-Hungary (Styria, Lower Austria and Northern Hungary), Greece, Canada (Quebec), Washington and California. The latter two sources represent chiefly the crypto-crystalline variety.

The chemical composition of the several types of magnesite is given in the following table:

Table 7

	Styria (calcined), per cent	Quebec (calcined), per cent	Washington (calcined), per cent	California (crude), per cent
Silica	2.8	5.5	6.8	3.86
Ferric oxide	7.1	7.0	5.0	0.80
Alumina	0.6	2.0	2.0	
Magnesium oxide	85.6	67.5	83.1	43.47
Lime	2.9	17.7	3.1	2.00
Carbon dioxide	....	....	....	49.48

In the manufacture of these refractories it is necessary to calcine the magnesite before it can be molded owing to the large shrinkage which the mineral undergoes and it is desirable to reduce it, as much as possible, to a condition where it will remain fairly constant in volume upon refining. Owing to the exceedingly high temperatures

required in the case of pure magnesite this operation would be impossible under commercial conditions were it not for the presence of impurities, especially iron oxide, which bring about a certain degree of softening or sintering. In the Austrian magnesite the iron occurs in the natural condition, a fact which has made these ores so desirable in the past. In using the purer minerals the iron oxide must be added and ground with the raw rock before calcination. This applies particularly to the American magnesites.

The amount of iron, in the form of hematite ore, employed for this purpose is sufficient to bring the composition to that of the Austrian rock. The calcination is conducted in rotary kilns. Magnesite bricks are usually made from dead-burnt magnesite, ground quite fine, and mixed with a little water in the wet pan. The molding is accomplished by means of a powerful press. The bricks are then dried and burnt in down-draft kilns, being boxed in with silica brick, to a temperature about cone 18. The shrinkage in the firing of the bricks is about 0.125 in. per foot.

For mortar or cement used in building up furnace bottoms the ground, calcined magnesite is used, dampened with a little water. In the calcination of the commercial magnesite the mineral periclase is produced, sp. gr. 3.674. The completeness of the calcination may be judged from determination of the density. In addition to the principal mineral constituent, periclase, there are formed also accessory minerals as magnesium ferrite, $MgO.Fe_2O_3$; forsterite, $2MgO.SiO_2$; olivine, $2(MgFe)O.SiO_2$ and monticellite, $CaO.MgO.SiO_2$. Magnesite of the purer grades has also been sintered or fused in the electric furnace and made into crucibles, tubes, etc., for laboratory use and research purposes. Thus, Burgess forms crucibles from crushed, sintered or fused magnesite, by electric heating in graphite molds, and Yensen by pressing pulverized, fused MgO with a small amount of magnesium hydrate in steel molds and heating the articles to 1,800°C. in an electric furnace.

Properties of Magnesite Refractories.—The chemical and mineralogical composition of the more impure materials has already been indicated. From the nature of these refractories it is apparent that they will take up moisture from the air so that, according to Howe and McDowell, 2.5 per cent by weight may be absorbed after moistening the material daily, after 30 days. This hydration becomes more and more prominent with increase in the lime content with the result that the material may crumble and disintegrate, if the CaO is present in larger amounts. Thus, burnt dolomite may absorb 25 or more per cent of H_2O. Magnesite refractories therefore should be kept in dry storage rooms.

The fusion point of pure magnesia has been estimated to be 2,800°C. but that of the impure magnesite of commerce is about 2,165°C., according to Kanolt.[1] Washburn computes the latent heat of fusion to be 700 cal. per gram.

Upon reheating to temperatures up to 1,500°C., an appreciable change in volume of up to 3 per cent may take place with well-burnt magnesite refractories. No figures seem to be available concerning the porosity of this material though it must be fairly high and between 25 to 30 per cent. The density of dead-burnt magnesite is about 3.5, that of fused MgO, 3.58. The bulk specific gravity of bricks is 2.6 to 2.75.

Magnesite bricks, subjected to a pressure of 50 lb. per square inch, seem to fail suddenly at from 1,450 to 1,550°C. through shearing. It has been suggested that this type of failure is due to a molecular change. It certainly cannot be ascribed to softening of the mass. Magnesite, however, becomes distinctly plastic at temperatures much below the fusion point.

The mean specific heat of pure magnesia is 0.234 at 61°C. and 0.2762 at 390°, mean temperatures. For magnesite bricks Heyn gives the values 0.208 at 0°C. and 0.291 at 1,300°, and Steger 0.225 between 20 and 200°C.

[1] *Tech. Paper* No. 10, Bureau of Standards.

The thermal conductivity has been reported by Dudley to be 0.0135 cal. per second through 1 c.c. per 1°C. temperature difference, between 445 and 830°C. On the other hand, Goerens and Gillis found this value to be 0.0117 between 100 to 200°C., 0.1 between 0 and 1,000° and 0.008 at 1,000°C. Dougill, Hodsman and Cobb found conductivity to decrease rapidly with increasing temperature and computed the mean value between 0 and 1,000°C. to be 0.0155. The conductivity at 1,000°C. was 0.0085.

The linear thermal expansion expressed in per cent of the original length and the average of the results of three observers is 0.24 per cent at 200°C.; 0.51 at 1,200°; 1.88 at 1,400° and 1.88 at 1,600°.[1]

Magnesite bricks do not resist sudden temperature changes well and show a decided tendency to spall under such conditions. This type of refractories in contact with clay is attacked vigorously by the latter at furnace temperatures. Silica materials attack it likewise though less violently. Clay and magnesite and silica-magnesite refractories must therefore be kept from coming in contact by means of a neutral parting such as chrome brick or cement. Carbon likewise reacts with magnesia at high temperatures as does phosphorus.

Magnesite bricks show in the cold state a compression strength of approximately 2,000 to 3,000 lb. per square inch. Their resistance to abrasion and impact is not marked owing to the comparatively weak bond. The electrical resistivity of magnesite refractories has been found by Stansfield, MacLeod and McMahon[2] to be 6,200 ohms per cm.[3] at 1,300°C.; 420 at 1,400°; 55 at 1,500°; 30 at 1,550° and 25 at 1,565°.

Dolomite Refractories.—The scarcity of magnesite during the war has brought about the marketing of highly burnt impure dolomites or crushed stone of this character treated with slag to make them more resistant to hydration. This material has been used quite successfully for open-hearth furnace bottoms but attempts to make from it bricks, corresponding to magnesite bricks, have not been fruitful. Dolomite refractories, while they may replace magnesite for furnace bottoms, cannot be said to equal the latter. Storage usually hydrates the lime to a more or less pronounced degree, thus lowering the mechanical strength of the product and in extreme cases bringing about its disintegration.

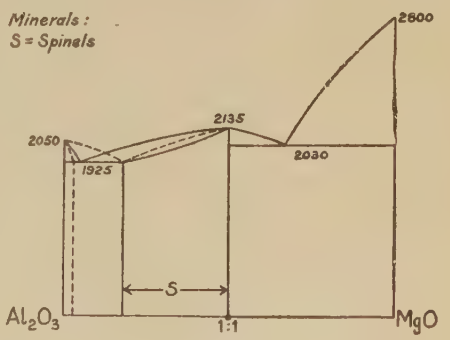

FIG. 11.—MgO-Al_2O_3 fusion diagram.

Spinel Refractories.—It is possible to produce a series of refractories consisting of magnesia and alumina combined to form the spinel, $MgO.Al_2O_3$ or containing an excess of either constituent. Such a combination reduces materially the sintering temperature required for attaining constancy of volume of the calcine, especially in the presence of such impurities as occur naturally in magnesite and bauxite. An inspection of the fusion diagram of the system $MgO.Al_2O_3$ (Geophysical Laboratory), given in Fig. 11, shows the fusion point of the spinel to be 2,135°C. and that of the eutectic between this compound and MgO to be 2,030°.

[1] From manuscript of HOWE and MCDOWELL, previous to publication in *Journ.* Am. Ceram. Soc.
[2] *Trans.* Am. Electrochem. Soc., Vol. **22**, p. 89 (1912

The slope of the curve is such that no decided drop in the fusion temperature occurs when the molecular proportion of the two constituents is changed within reasonable limits. It is evident, therefore, that the spinel offers a type of refractory of very promising usefulness, especially under conditions requiring basic refractories.

In producing this refractory the purer grades of magnesite and bauxite should be used and no binder such as clay can be employed. Wherever possible an organic binder like linseed oil or a small amount of low-burnt magnesia may be used. The Bureau of Standards has made crucibles of magnesia-spinel successfully without the addition of any inorganic binder by grinding a portion of the material very fine. According to the purity of the magnesite and bauxite the calcining temperature should vary from cone 20 to 30 and the firing of the articles must be carried to at least cone 20. The required proportion of magnesite to bauxite corresponds to the ratio 84 : 102 referred to the $MgCO_3$ and Al_2O_3 contained in these materials.

Carbon Refractories.—Clay materials containing carbon in some form have been in use for a long time. Carbon itself must be considered a refractory of high grade which can be used wherever the possibility of continuous oxidation is excluded.

Brick made from a mixture of crushed coke and clay or from coke bonded by means of tar have been made many years ago. These materials are usually hand molded, dried and fired in muffles with as complete exclusion of air as possible. The carbonization of the tar cements together the coke particles to a compact mass. The coke may also be replaced by graphite. Refractories of this type combine good heat-resisting power with excellent thermal conductivity.

Mixtures of clay and graphite are used very largely in the crucibles employed in the melting of metals. Here the graphite contributes toward the refractoriness and heat conductivity of the mass, prevents the oxidation of the metal, and produces a smooth surface for pouring.

The function of the clay is that of a bonding material, which makes possible the shaping of the crucible and the cementing together of the graphite flakes. At the same time, it covers the particles, and thus protects them from oxidation. In order to be most effective in regard to this point, it is necessary that the clay contract and become dense at as low a temperature as is consistent with the required refractoriness.

As to the real nature of graphite we are still in the dark. It is crystalline, rhombohedral, and stable at the highest temperatures. Its great softness, flaky structure, and feeble metallic luster are characteristic. Its specific heat varies in the natural and artificial varieties and according to the purity of the materials. Values have been determined for Ceylon graphite of from 0.174 to 0.2019 and for blast-furnace graphite of from 0.166 to 0.1970. The specific heat increases with temperature and has been found to be 0.467 at 977°C. The coefficient of thermal conductivity of graphite (gram calories per degree Centigrade through 1 $cm.^3$) is 0.0141, and hence, five times that of burned clay. The linear coefficient of expansion is 0.00000786 at 40°C. The excellent electrical conductivity of graphite is well known.

The principal foreign sources of graphite are: Ceylon, Canada, Mexico and Madagascar. In the United States the chief sources of the mineral are to be found in Alabama, New York and Pennsylvania. Artificial graphite is produced in the electric furnace, and a soft variety known as "kish" is the product of the blast furnace. Possible sources of carbon which may be used in this connection are retort graphite formed in gas retorts and the carbon resulting from the decomposition of hydrocarbons.

In European practice both coke and retort graphite have been admixed with the natural graphite for making crucibles. The practice is not desirable for steel melting,

since the amorphous forms of carbon are more soluble in the molten metal than crystallized graphite, and hence cause too much cementation. Their more rapid rate of oxidation is likewise detrimental for this purpose.

Some of the Birmingham crucibles are said to be made up of 3 parts (by weight) of graphite, 2 of hard coke, 2 of Stourbridge clay, and 1 of ground sherds from old crucibles. Crucibles used in a German foundry have been made from a mixture consisting of 5 parts of ground coke, 4 of graphite, and 8 of Stourbridge clay and grog. If the necessity arose, it would be, of course, possible to use crucibles in which all the carbon is introduced in the form of retort carbon, coke, or electric-furnace graphite, even though the results were not as satisfactory as those obtained with the natural graphite.

Crucible Mixtures.—The main requirements of a suitable crucible mixture are that it must possess satisfactory working qualities, good strength in the dried state, ability to withstand sudden temperature changes, good thermal conductivity, freedom from pinholes, sufficient mechanical strength to stand up under rough treatment with tongs, and slow oxidation of the graphite.

The clays most desirable for this purpose are those of the dense-burning type, which soften slightly at the heat of the brass furnace. The open-burning clays are not suitable for this purpose, though they may be used for the purpose of stiffening clays deficient in refractoriness. The clays to be considered here are those of the Klingenberg type; some of the English ball clays, such as that marketed as "Dorset;" the ball clays from Mayfield, Ky., and the Paris, Tenn., district; the bonding clay mixtures marketed at St. Louis, Mo.; the plastic bond clays from northern Mississippi, and Anna, Ill. In every case those clays are to be preferred which vitrify close to 1,150°C.

The composition of graphite crucibles varies considerably with reference to whether they are to be used for brass or for steel melting. Some of the mixtures which have come to the knowledge of the writer are as follows, the first four being principally intended for melting brass and the last two for steel:

TABLE 8

	Per cent					
	a	*b*	*c*	*d*	*e*	*f*
Graphite	48	57.5	55	50	58	55
Bond clay	32	25.5	35	40	35	30
Kaolin, plastic	6	10.4	5	..	..	7
Sand	..	....	..	5	7	8
Crushed pot shell	14	....	5	5		
Potter's flint	..	6.6				

The addition of sand can be avoided by the use of siliceous clay, like that from Arkansas (Lester), which makes possible a reduction of the clay content, if this is desirable, and yet the maintenance of good working quality.

Manufacture and Use of Graphite Crucibles.—The preparation of the graphite consists usually in grinding it by means of a burrstone mill or pan or disintegrator and sizing it by means of sieves. A smaller amount of the dust is used, which is frequently utilized for nozzles, stoppers, and other articles. The sized mineral is

usually stored in bins, from which it is weighed out in the proper proportions, together with the ground-clay mixture. The materials are then mixed together in the dry state, avoiding further grinding, for which purpose a kneading machine may be employed. As soon as the dry mixing is completed water is added and the tempering continued until the soft plastic state has been attained and the whole mass is of uniform consistency throughout. The material is then allowed to cure or age for some time in the clay cellar. The crucibles are molded by means of the potters' jolly in plaster or metal molds lined with cloth. As regards to drying, the usual pottery practice is followed. The application of the modern humidity-drying methods, at least for the larger crucibles, should offer an opportunity for improvement, since uniformity of shrinkage is not always secured under the present conditions.

The firing of the crucibles is carried to a temperature but little above the temperature of complete dehydration of the clay, usually between 650 to 700°C. Even in muffle kilns some loss of graphite occurs on the surface of the crucibles through oxidation, as is evidenced by the white coating of clay.

It is quite apparent to the casual observer that the treatment to which graphite crucibles are subjected is often extraordinarily severe. In the first place it is doubtful whether the function of drying out the crucibles is properly understood. Graphite crucibles are not fired to a high initial temperature and hence retain a porous structure. In common with all other clay products not burned to vitrification the crucibles are hygroscopic; that is, they greedily absorb moisture from the atmosphere, which is more difficult to expel than we generally realize.

Low-fired clays differ widely as to their hygroscopic capacity and the temperature at which they release the moisture thus absorbed. At the same time, the expulsion of this moisture requires considerable time. Therefore, it is not sufficient simply to keep the crucibles in a warm place, but they should be finally maintained for at least several days at a higher temperature, say 150°C., before being placed in the furnace. In Europe the crucibles on being taken from this kiln are sometimes covered with a waterproof coat, such as tar or pitch dissolved in turpentine.

The preheating of the crucible just before putting into the heat for first time, likewise, is often too abrupt, and hence the temperature change to which they are subjected too violent. The ideal method of handling the fresh crucibles would be to place them in a special furnace kept heated by the waste gases of the melting furnace, where the temperature could be brought up uniformly to somewhat below red heat.

The conditions of heating are of great importance. It is evident that the intense heat of an oil burner will bring a large crucible far closer to the softening point than when it is exposed to the fire of a coke furnace. The larger the crucible and the higher the level of the liquid metal, the lower in temperature must be the failing point of the crucible, since at furnace temperatures even comparatively small loads tend to bring about deformation. Add to these factors the effects of unequal heating and rough handling, and it can readily be seen why the life of crucibles varies so widely.

Silicon Carbide Refractories.—The use of silicon carbide, SiC, as a refractory is becoming more and more extensive. The material may be shaped without the use of a binder or with one such as clay or tar. In the former case cementation is brought about by the interlocking of the crystals in refiring in the electric furnace. In the second the strength must necessarily depend upon that of the bonding material and the refractoriness is limited by the latter, as well. Silicon carbide is being used quite largely now in crucibles, in which it displaces graphite.

The crystalline silicon carbide is produced at about 1,840°C. and dissociates at about 2,240°C. Fusion or softening does not occur below this point. Under actively oxidizing conditions decomposition undoubtedly occurs to a certain extent. But oxidation goes on slowly when in contact with gases containing but a small amount of free oxygen. For many purposes these refractories are exceedingly useful owing to their high thermal conductivity, their low coefficient of expansion and their rigidity and mechanical strength. The specific heat is given as being 0.162 for the material bonded by the cementation process and 0.180 for the one bonded with clay. The thermal conductivity of the former is 0.0275 and for the latter 0.0243. The conductivity of the silicon carbide refractories is therefore about seven times that of clay firebrick. The coefficient of thermal expansion is 0.0000047 per degree Centigrade, which is lower than that of clay materials. The porosity of silicon carbide products varies between 15 to 20 per cent and the bulk specific gravity from 2.1 to 2.5. The material resists load conditions at furnace temperatures (1,400 to 1,450°C.) without any appreciable deformation.

Special Refractories.—Under certain conditions refractories of special qualities may be employed such as zirconium oxide, zirconium silicate, chromite, fused silica, boron nitride, aluminum nitride, lime, beryllium oxide, cerium dioxide, thoria, asbestos and various synthetic combinations.

Zirconia.—The sources of this oxide are the silicate, $ZrSiO_4$, zircon, baddeleyite, containing on an average 84 per cent of ZrO_2, and the monazite sands. Pure zirconia fuses at about 2,600°C. which is depressed by the presence of silica, iron, etc. The zirconia must first be calcined at as high a temperature as possible, up to 2,000°C., according to the amount of impurities present, then ground and molded either by pressure, by hand molding or by casting. In the first two processes an organic binder like linseed oil, starch, etc., may be employed; in the latter water is used and the procedure is like that of ceramic casting. From the crude zirconia or zircon, iron and some silica may be eliminated by treatment with chlorine at about 800°C. The thermal conductivity of ZrO_2 is quite low and the coefficient of expansion approaches that of quartz glass, being about 0.00000084. It resists fused alkalies very satisfactorily. Pure zirconia has a specific electrical conductivity of 0.0008 reciprocal ohms at 1,200°C. and 0.0034 at 1,400°. The addition of alumina increases the conductivity. Thus one molecule of Al_2O_3 added to nine of zirconia increases the conductivity at 1,200°C. to 0.00255 reciprocal ohms. Cerium oxide, used in the same proportion, gives a conductivity of 0.0075 reciprocal ohms at 1,000° and ferric oxide 0.0358 at 1,287°. Zirconia refractories therefore conduct the electric current when heated. Thus the pencils of the Nernst lamp containing about 85 per cent of zirconia and 15 per cent yttria become conductors, a principle which has also been employed in the construction of electric furnaces.

Through fine grinding in water the zirconia or zirconium silicate may be reduced to a state where it may be molded without the addition of cementing materials. As in the case of all highly refractory oxides the addition of such silicates as clay should be avoided owing to the great depression of the melting point. The contraction of zirconia in calcination may be accelerated through the use of a small amount of boric acid.

Zirconia combines with carbon to form a carbide which is also an abrasive. Combining zirconia with small quantities of silica, beryllia, magnesia and alumina lowers the melting point (alumina being the least detrimental), while thoria and yttria raise

it. The lack of data available upon the systems comprising zirconia and the various oxides makes it difficult to make predictions as to the most suitable combinations, but it is evident that there are compounds particularly well suited for specific requirements. In the production of zirconia refractories the chief difficulties lie in the proper molding of the feebly plastic mass and in the large firing shrinkage which can be reduced only by previous calcination at a high temperature. The shrinkage factor is eliminated where it is possible to use $ZrSiO_4$.

Chromite refractories are employed in furnaces as partings between the silica and magnesite bricks and hence are considered neutral. They consist principally of crushed and ground chromite, and bricks are molded from this material which may be bonded together with a small amount of clay or burnt magnesia by pressing under a heavy pressure. The bricks are then dried and fired to a temperature of about cone 16. The composition of these refractories fluctuates between the following values:

	Per Cent
Chromium oxide	38.0 to 40.0
Alumina	24.5 to 40.0
Ferric oxide	17.5 to 40.0
Silica	3.0 to 40.0
Magnesia	15.0 to 40.0

Chrome refractories are excellent conductors of heat. Little information is available concerning their physical properties. They possess a fairly high coefficient of thermal expansion and evidently undergo molecular changes upon reheating. Chrome brick resist load conditions quite well but are apt to fail suddenly at about 1,450°C. Their use in metallurgical practice has decreased considerably during the past few years. Chrome cement for partings is, however, still being applied.

Fused silica is used as a refractory in pyrometer tubes, crucibles, etc., where its limitation is the change in structure due to crystallization.

Boron and aluminum nitrides are materials suitable for high temperatures but are subject to decomposition and are not used to any extent.

Lime is a basic refractory of the highest type limited in its use through its hydration in the cold state. It requires exceedingly high temperatures for its calcination to constant volume.

The oxides of beryllium, cerium and thorium, etc., are precluded from extensive use owing to their cost. Their employment in gas mantles is too familiar to need further discussion.

Asbestos and other magnesium silicates are sometimes used as refractories. Their low melting point, however, excludes them from high-temperature work.

In addition, many combinations between alumina, magnesia and silica are possible which may render good service under special conditions. As an example, a composition suitable for spark plugs and high-tension insulators used at about red heat, as in the Cottrell process, is as follows:

	Per Cent
Calcine:	
Kaolin	75.10
Magnesite	8.77
Alumina	16.13
Body:	
Calcine	60.00
Kaolin	30.00
Ball clay	10.00

This refractory must be fired to the temperature corresponding to cone 17 and cooled quickly so as to produce a dense structure consisting to a large extent of small crystals of sillimanite. The calcine requires to be fired to the same temperature as the body.

Insulating Refractories.—For the purpose of preventing the large losses of heat occurring through the walls and arches of furnaces the use of insulating materials applied to the outside or inside of the walls is becoming more and more extensive. The substances used for this purpose are principally kieselguhr or diatomaceous earth or a combination of this material pulverized and mixed with cork dust and bonded by means of plastic clay to form bricks. These refractories are not intended for exposure to high temperatures and cannot be used under such conditions owing to their large contraction in volume and also the low softening point of the kieselguhr-clay combination. The insulating materials of this type serve their purpose admirably and deserve general application for the prevention of the enormous heat losses of industrial practice. Their chief advantage lies, of course, in the low thermal conductivity which is about 0.000344 in c.g.s. units or 1 B.t.u. per hour, per square foot, through 1 in., or about one-sixth of the conductivity of the average fire brick at 200°F. This reduction becomes still more marked at higher temperatures so that at 1,600°F. it may be only one-ninth of the thermal conductivity of the refractories commonly used. The temperature coefficient of these materials is close to unity so that the conductivity remains quite constant until temperatures are reached which bring about a change in structure *i.e.*, a decrease in the porosity. The light weight of the insulating materials, 25 to 30 lb. per cubic foot, is also a desirable feature. They are composed practically of silica. In commercial use the materials are marketed in the form of blocks or bricks or as a powder.

For uses where resistance to higher temperatures is unnecessary and where some of the insulating quality may be sacrificed compositions consisting of approximately 49 per cent of cork (20 to 60 mesh), 16 of plastic fireclay, 22 of Georgia or Florida kaolin and 13 of crushed firebrick, 4 to 16 mesh, may prove useful.

In using heat-insulating materials it is necessary to realize that insulation necessarily raises the mean temperature of the wall between the surface exposed to the heat and the insulation to a point far above that applying to ordinary conditions. The damming up of the heat thus raises especially the temperature of the surface of the firebrick. It is inevitable therefore that insulation requires the use of materials which are more heat resisting. Many instances are on record where firebrick have failed soon after insulation was applied. Heat insulation may thus be said to have increased the necessity for superior refractories.

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660.L71 V01

a39001 007491023b

S0-ELH-381